APPRAISAL OF COAL MINE VENTILATION

A REPORT

BY

NORWEST RESOURCE CONSULTANTS LTD.

IN ASSOCIATION WITH

DAMES AND MOORE

Work on this project was conducted under
the auspices of the Canada Centre for Mineral
and Energy Technology, Energy Mines and Resources, Canada

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NorWest

Norwest Resource Consultants Ltd.

Suite 600, 610 - 8th Avenue S.W.
Calgary, Alberta T2P 1G5
Telephone 403-237-7763

September 27, 1982

Scientific Authority
Dr. R.N. Chakravorty
Department of Energy Mines and Resources
CANMET, Mining Research Laboratories
3303 33 Street N.W.
Calgary, Alta.

Our File: 8.3.8125

Dear Sir:

Ref: D.S.S. File No. 18SQ.23440-0-9098

With this letter we enclose the final report "Appraisal of Underground Coal Mine Ventilation". Norwest Resource Consultants were contracted by the Federal Department of Supplies and Services (D.S.S.) to conduct the investigations, make the appraisals and submit the report to you as the appointed Scientific Authority.

In carrying out this work, Norwest has worked in association with engineers of Dames and Moore of Cincinnati who were supervised by Mr. D.J. Steele. Overall direction of the project was by Mr. H.G. Stephenson of Norwest.

The cooperation and suggestions offered by engineers of the coal mining companies and of Provincial Regulatory Agencies concerned with mine ventilation, have been greatly appreciated. The response of coal engineers and technical personnel to our investigation indicated a growing interest in the application of a more scientific approach to solving coal mine ventilation problems. It is our belief that this report will stimulate this interest and that our recommendations for further investigation and research will be endorsed by the industry. It is this growing interest in the scientific approach to ventilation that confirmed our views on the value of the bibliography which forms a sizeable part of this report. It became evident that the time spent in compiling and checking an updated and comprehensive bibliography was well worth while.

We sincerely hope that you will find the report meets all your requirements.

Yours truly,
NORWEST RESOURCE CONSULTANTS LTD.

H.G. Stephenson
President

Canmore
P.O. Box 449,
Canmore, Alberta T0L 0M0
Telephone 403-678-4311

Utah
Suite 606, #10 Exchange Place
Salt Lake City, Utah, U.S.A. 84111
Telephone 801-364-5343

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1. GENERAL INTRODUCTION

1.1 SCOPE OF WORK

The scope of work for this report was set out as "Terms of Reference" in a communication from the Federal Department of Supply and Services; D.S.S. File No: 18SQ.23440-0-9098. Title: "Appraisal of Coal Mine Ventilation".

The work description was outlined as follows:

The Department of Energy, Mines and Resources has a requirement to:

1. **Evaluate ventilation survey techniques and their potential applications in Canadian coal mines. Discuss computer program development and availability for planning, design and analysis of coal mine ventilation systems.**

2. **Summarize the state of the art regarding the choice and applications of main fans and booster fans in underground coal mines.**

3. **Evaluate the ventilation aspect of methane control techniques in high productivity coal mines keeping in view Canadian requirements. In particular, the study should include strategies for ventilation of advance and retreat longwall faces. Emphasis should be given to problems associated with longwall face ventilation in large and extensive operations.**

4. **Discuss the "Cundy" system of ventilation as practiced in Canadian coal mines and evaluate its effectiveness under actual mining conditions.**

5. **Evaluate the procedures for building ventilation stoppings in Canadian coal mines and identify areas where research and development can be considered necessary.**

6. **Discuss the interaction between mine fire and mine ventilation systems as applicable to Canadian coal mines. Evaluate ventilation strategies to control leakage of air in underground coal mines.**

7. **Evaluate the present ventilation practices in Canadian coal mines by visits and discussions with mine officials and provincial inspectorate.**

The study should cover longwall, room and pillar, and hydraulic mines.

Identify areas where research and development work will prove useful.

1.2 APPROACH TO THE TASK

The approach taken, to carry out the tasks assigned in the terms of reference, was as follows:

(a) An overview of technical papers and current experience was conducted. Much of this information was already published and known to the various contributors to this report. Some material was referred to by technical personnel during mine visits and some was encountered during our research.

(b) Vists were made to operating underground coal mines and discussions held with engineering and operating personnel. The operations visited were:

(i) In Nova Scotia:
No. 26 Colliery (C.B.D.C.)
Lingan Mine (C.B.D.C.)
Prince Mine (C.B.D.C.)

followed by meetings with technical personnel at the Cape Breton Development Corporation (C.B.D.C.) headquarters, Glace Bay.

(ii) In Alberta a visit was made to McIntyre Mines Ltd. at Grande Cache; the development drifts at Cardinal River Coals and the Atlas Mine at East Coulee.

(iii) in British Columbia, the hydraulic and conventional underground operations of B.C. Coal were visited.

(c) The consultants discussed the subject of this report with government organizations in Nova Scotia, Alberta and British Columbia.

(d) The consultants' staff made full use of their experience in Canada and other countries in preparing this report.

At all times the stated or apparent needs of the operators for research and development of new and improved mine ventilation techniques was kept in mind.

1.3 FORMAT OF REPORT

The report is presented in two parts. Sections 2 to 7 inclusive form Part I of the report which deals with current mine ventilation technology, concentrating on those subjects itemized in the Terms of Reference. Sections 8 to 11 inclusive form Part II which describes current Canadian coal mine ventilation practice and includes recommendations for research and development. A bibliography is included.

It is obvious that this report could not be, and was not intended to be, a compendium of all modern coal mining ventilation technology and techniques. A general overview of current techniques and trends is provided with appropriate references for those seeking more detailed information.

The recommendations for Research and Development are believed to reflect the practical needs of the industry now and in the foreseeable future.

1.4 ACKNOWLEDGEMENTS

The authors wish to record their grateful thanks to the coal operators, government resource and mine inspection departments, and the many individuals in the Canadian Coal Mining Industry who gave their cooperation, advice and comments during the preparation of this report.

PART I

CURRENT COAL MINE VENTILATION TECHNOLOGY

2. VENTILATION SURVEYING AND MONITORING

2.1 INTRODUCTION

The general objectives of ventilation surveying and monitoring are to permit efficient control of air distribution in mines, to maintain a safe working environment, and to provide a basis for the planned provision and maintenance of adequate ventilation as mines develop. Surveys may be directed to more specific objectives, such as sources and quantities of methane emissions, as well as the environmental problems of dust, heat, and humidity. Attainment of these objectives requires the systematic collection of qualitative and quantitative data which may be used for a number of purposes, e.g.:

- to determine air distribution and to identify methane emission sources
- to locate leakages
- to calculate ventilation power costs
- to detect abnormal ventilation conditions, incidents, and heatings
- to determine optimum locations for booster fans
- to design the ventilation system to comply with site-specific requirements and to meet future ventilation needs

2.2 QUALITATIVE DETERMINATIONS

The routine sampling of mine air at strategic points has been required by regulatory authorities for many years and has come to be regarded as the main criterion for measuring the efficiency of ventilation. Regular sampling of the mine air for analysis, in addition to pre-shift and on-shift determination of methane content during underground inspections, has been the norm. The flame safety lamp has largely been superseded by hand-held methanometers and portable, automatic, methane detectors. Hand-held methanometers usually consist of a sensor, and indicating meter, alarm circuits and indicators which are operated by a rechargeable battery. There are no provisions for incorporating the instrument into any external monitoring circuitry.

The replacement of the flame safety lamp by the hand held methanometer is taking away from the front line supervisor and those who examine mine sections on a day to day basis, the ability to detect a lack of oxygen. Oxygen deficiencies are not uncommon under certain circumstances in the mountain mines of Alberta at the high elevations (say above 1500 m.).

Spot sampling has limitations in that trends towards deterioration of the ventilation system either in general or in specific terms may go unnoticed (Swift, 1965). The samples may not be representative of the gas emission conditions in a working district of a mine and it is unlikely that sources of gas emission will be detected. The possibility exists that local concentrations of methane may accumulate without indication in the general body ventilation. Accordingly, static methane monitors have been brought into use which provide continuous monitoring and recording as well as visual indication of danger levels. The principles of pellistor element monitors have been described by Baker and Windle (1969) and current developments are outlined by Miller (1981). In general terms, static methane monitor readings are accurate to ± 4 percent and provide alarm indications when the methane concentrations exceed a pre-set figure in the range of 0.5 to 2.5 percent of methane.

The detection of spontaneous combustion is by methods based on changes either in temperature or in air composition. The development of the infrared thermometer (Chakravorty and Feng, 1978) enables areas to be scanned for evidence of temperature increases. Systematic surveying with the instrument can allow trends to be developed and assessed. Detection of changes in air composition have involved the use of carbon monoxide as an indicator. Professor Graham (1930) showed how the ratio of carbon monoxide present to oxygen deficiency (compared to the proportion of oxygen found in normal fresh air) is independent of changes in the ventilation airflow. Continuous monitoring of carbon monoxide is internationally recognized as the prime method for early detection of spontaneous combustion and experience has been gained with monitors of the Cora and Ecolyser types (Goddard, 1977).

Experiments with Taguchi-type gas sensors have been described (Criddle, 1981) and it was considered that they may prove useful in the early detection of heatings and fires.

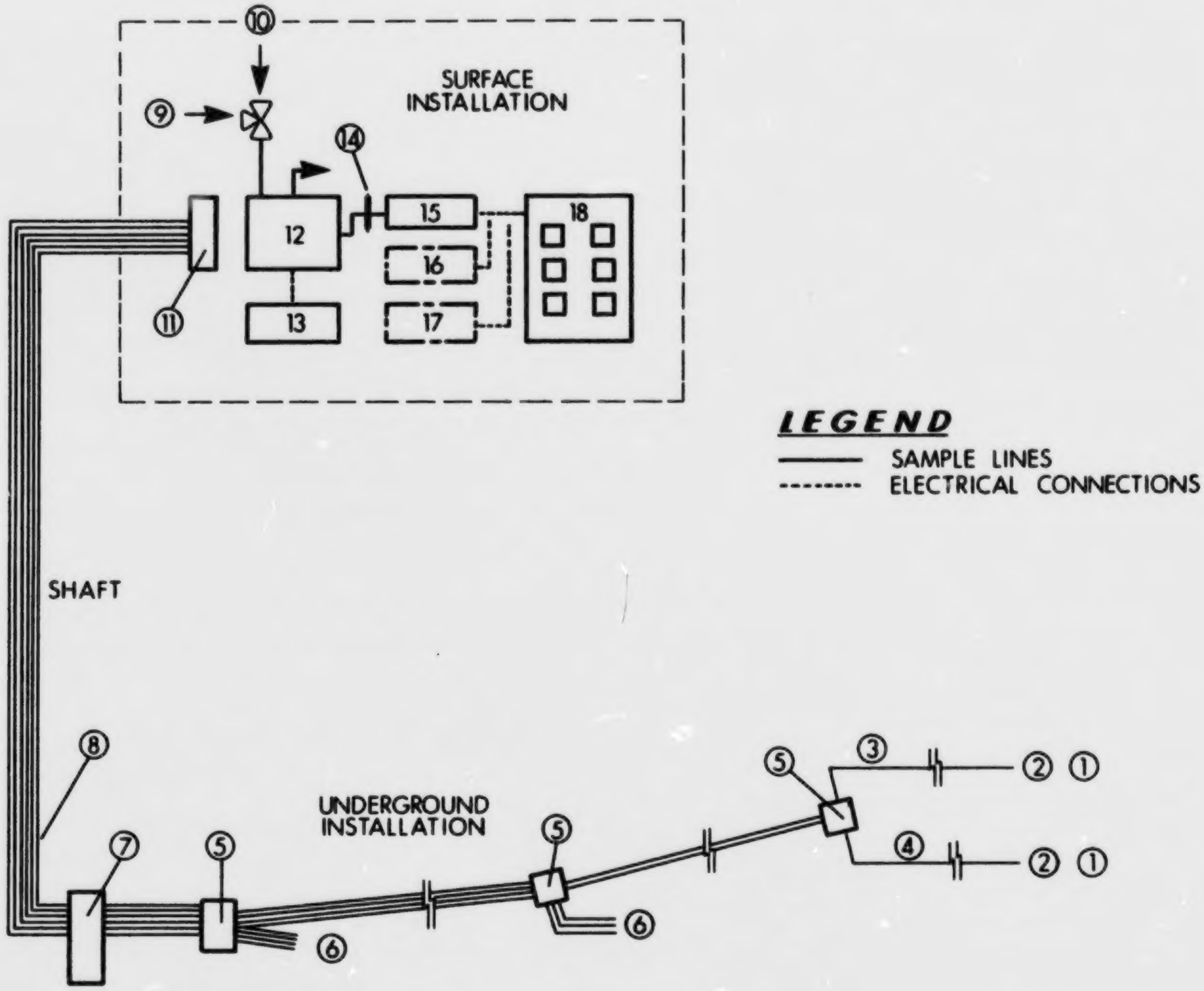
The qualitative survey of a mine's ventilation system has been developed to provide monitoring of the whole mine with locations in the circuits being continuously monitored (Chamberlain, 1973). The technique enables mine air samples to be brought continuously to the surface from various parts of the workings for analysis.

This technique, known as the tube bundle system (NCB, 1977), is shown on Figure 2.1. The system consists of multi-core tubes, usually up to 12 in number, housed within armoured hose and suspended through mine roadways to the selected sampling sites. The tubes are channelled to a surface extractor pump adjacent to a gas analyser house. Each sample is analysed for carbon monoxide, carbon dioxide, and methane by the infrared absorptiometric method and for oxygen by a paramagnetic method. Nitrogen is deduced by difference. The system is operated by a micro-processor which is programmed to calculate the Graham ratio and oxygen deficiency. The system can handle samples at the rate of one per minute, though the actual sampling time depends also upon the distance from the sampling point to the surface. Sample travelling time is of the order of 80 meters per minute. The gas sample is first cooled and then passed through the gas analyser. Accuracy of the analysers is maintained by an automatic zeroing system that operates at the start of each sampling sequence and, in addition, known test samples are processed twice per day.

The tube bundle system provides a full air analysis for the selected points. Analysis on the surface has two main advantages:

a) Sampling through the tube bundles can always be maintained, even when the supply of power to a district is cut off and men are withdrawn.
b) The system can monitor a large number of points and can be extended almost indefinitely.

A major disadvantage is the slow response time which makes it unsuitable for outburst situations. This disadvantage can, to some extent, be overcome by


⑩
SURFACE INSTALLATION
⑨
⑭
12
15
18
16
17
13
⑪
LEGEND
——— SAMPLE LINES
-------- ELECTRICAL CONNECTIONS
SHAFT
⑧
⑦
⑤
UNDERGROUND INSTALLATION
⑥
⑤
⑥
⑤
③
② ①
④
② ①

1) DUST TRAP(S)
2) WATER TRAP
3) INTAKE AIRWAY
4) RETURN AIRWAY
5) JUNCTION BOX
6) BRANCH TO OTHER DISTRICT(S)
7) BANK OF WATER TRAPS
8) ARMOURED TUBE BUNDLE IN SHAFT
9) STANDARDISATION GASES
10) FRESH AIR
11) WATER TRAPS OR HUMIDIFIER
12) CONTROL BOX
13) TIMING CONTROLS
14) UNOR FINE DUST FILTER
15) UNOR CO ANALYSER
16) OTHER ANALYSERS
17) OTHER ANALYSERS
18) DISPLAY CABINET

SCHEMATIC LAYOUT OF A TUBE BUNDLE INSTALLATION

FIGURE 2.1

SOURCE: NATIONAL COAL BOARD, 1972

moving the analyser underground and telemetering analysis results to the surface. This reduces the response time and allows the system to be used as a general fire alarm whereas surface analysis is only suitable for spontaneous heating alerts. A balance of the relative advantages and disadvantages for any particular mining situation must be made before final choice of system is determined.

Developing technology has made a reality of a computerized remote control and monitoring system (Corbett, 1979). One such data transmission is depicted in Figure 2.2. These systems consist of three main elements:

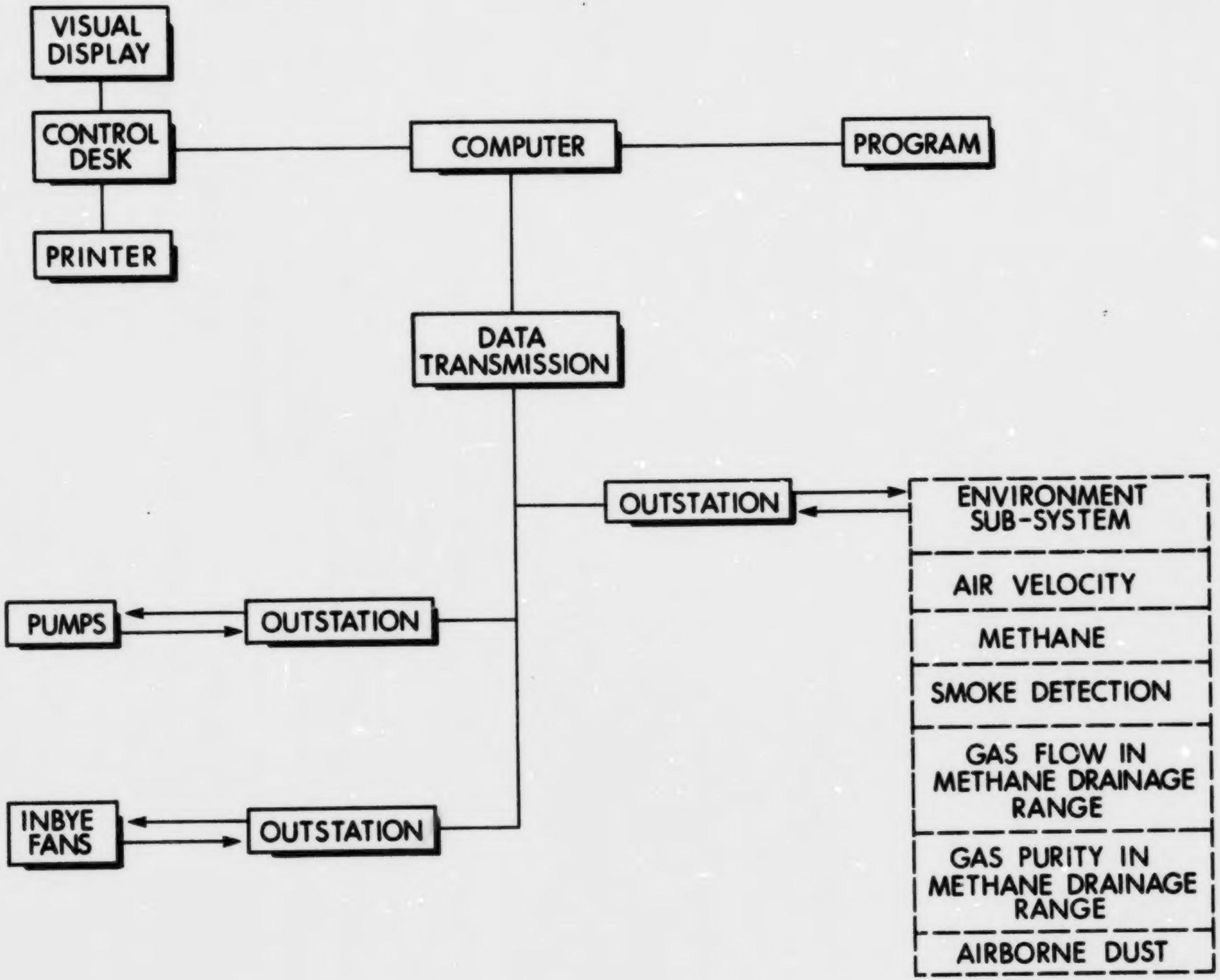
a) Out stations, at selected points underground, to receive signals from transducers which monitor the ventilation

b) A surface primary computer which scans the out stations in a predetermined order by time division multiplexing. Information fed back from these stations is recorded on a printer. An integral part of this computer is the control console with a function keyboard and twin monitor screens for graphic display. Also available are remote display terminals for use away from the central station

c) A secondary computer, of greater capacity than the primary, for long-term storage of data and feedback of information upon demand

Normally, production districts and development drivages will be monitored for:

- airflow; using a transducer with an instrument having three suitable velocity ranges
- full air analysis
- gas flow in the methane drainage range
- gas purity in the methane drainage range
- suction in the methane drainage range
- smoke detection
- combustible respirable dust levels
- effective temperature
- noise level indication


VISUAL DISPLAY
CONTROL DESK
PRINTER
COMPUTER
PROGRAM
DATA TRANSMISSION
OUTSTATION
ENVIRONMENT SUB-SYSTEM
AIR VELOCITY
METHANE
SMOKE DETECTION
GAS FLOW IN METHANE DRAINAGE RANGE
GAS PURITY IN METHANE DRAINAGE RANGE
AIRBORNE DUST
PUMPS
OUTSTATION
INBYE FANS
OUTSTATION

VENTILATION DATA TRANSMISSION SYSTEM

FIGURE 2.2

Additional functions, upon demand, would be calculation of the Graham ratio and visual display of the position of the sample in relation to the Coward triangle.

The transducers can be sited at selected sampling points. The layout for a typical production district has:

- an outstation sited at intake entrance to district; monitoring selected from the list above
- an outstation in primary working headings; monitoring selected items from the list above
- an outstation sited at the outer end of the district return; provided with full range of monitoring facilities

In the development drivages items are selected as appropriate.

Visual display units on the central control desk are the focal points for information fed back into the system. The normal display is a tabloid showing the monitoring point identified by district name and location and the status of the various parameters being monitored. The tabloid represents the up-to-date underground condition and forms a general mine display. Other programmes are displayed, as required, by typing the appropriate code on the function keyboard. The individual district display repeats the parameters covered by the general mine display, together with district additional functions. Graphics can be established to display current shift, previous shift, and previous 7 days. In effect, they display trends which are on instantaneous recall.

One of the main uses of the system is the early recognition and indication of abnormal environmental conditions. The computer is programmed for parameter norms. Movement away from these norms is indicated by a series of alerts followed by alarm should the adverse trend persist. With the analogue transducers, several levels of alert can be indicated before alarm is reached. With the digital transducer, only an alarm condition is displayed as there is no graded circumstance.

All continuous monitoring systems have the advantage of providing early warning of abnormal developments in the environment and so allow for early action to correct the situation. As such, they are of particular value in combating the development of spontaneous heatings.

2.3 QUANTITATIVE DETERMINATIONS

Quantitative assessment is primarily required to determine the existing distribution of the air, to locate leakages, and to enable calculation of the resistance of various sections of the ventilation system in conjunction with pressure determinations.

The normal method of quantity determination is to measure the velocity of air flow with either an anemometer or velometer and to determine the cross-sectional area of the sampling point. Calibration of the equipment has recently been improved by the utilization of the laser velocimeter and pitot-static tube (Haney, 1979).

Continuous monitoring of volume airflow has been made possible by development of velocity monitors of the swinging gate, pitot tube, grid and venturi types (Goddard, 1977). These are used in conjunction with other telemetric devices to permit the remote control and monitoring of booster and auxiliary fans. Currently two main types of air velocity monitor are used. The anemometer type incorporates a vane assembly with a means of detecting the change in capacitance as each vane passes a fixed point. The frequency of the pulses is converted into a d.c. analogue signal and provision is made, through damping of the responses, for the instrument to give an indication of mean air velocity. The monitor has a range of 0 to 10 m/s in three stages and readings are accurate to ± 6 percent. The ultrasonic vortex sensing monitor incorporates a strut in the path of the airstream which causes vortices behind the strut at a frequency corresponding to air velocity. The vortices are detected by ultrasonic transducers and converted to d.c. analogue signals. The range is 0 to 10 m/s in three stages and the reading accuracy in the uppermost range is ± 0.3 m/s. Neither of these instruments is intended for use in a duct or other restricted area. The use of telemetry through intrinsically safe circuits permits monitoring data to be transmitted to the surface.

The determination of very low quantities of airflow found in leakage paths and gob migration may require the use of tracer gases such as helium, nitrous oxide, or sulphur hexafluoride. The tracer gas chosen for the mining environment must be:

- non-toxic; even at relatively high concentrations;
- physiologically hazard free to persons exposed to the gas;
- insoluble in water;
- thermally stable and chemically inert;
- unadsorbed on the surface of any materials likely to be found underground;
- not occurring naturally in the environment under investigation.

Nitrus oxide has been used in gas tracer experiments underground (Higgins and Shuttleworth, 1958; Shuttleworth, 1967; Dunmore, 1969). Using infra-red gas analysis it was possible to determine low concentrations of the gas of the order of 0.01 percent (100 p.p.m.) to an accuracy of ± 1 percent of the readings with a lower limit of detection of about 0.0001 percent (1 p.p.m.). The disadvantages of nitrous oxide are that it may possibly be adsorbed by coal and soluble in water (Dunmore, 1969) and not wholly free from physiological effects (Wann, et al., 1977).

Helium has also been used as a tracer gas (NCB Internal Report, 1963). Using gas chromatographic methods of analysis and a katharometer detector its limit of detection (about 1 p.p.m.) was found to be similar to nitrous oxide.

The use of sulphur hexafluoride (SF_6) as a tracer gas in U.S. mines was first reported in 1974 (Thimons, et al., 1974). It has subsequently been used in a number of applications (Vinson and Kissel, 1976; Wann, et al., 1977). Sulphur hexafluoride meets practically all the requirements for a tracer gas to be used in coal mines. Using a gas chromatic method of analysis, with an electron capture detector, it is possible to detect 0.000005 p.p.m. The gas does not occur naturally in the mining environment but may be used in transformers and electric switchgear and it is therefore prudent to check whether electrical equipment of this nature is installed where the use of sulphur hexafluoride may be contemplated. Normal gas sampling equipment may be used but aerosol grease removers should not be used on gas sampling pumps otherwise difficulties will be encountered when the gas samples are analyzed.

Techniques employed use either pulse release or steady release of the tracer gas. Pulse release is preferable if the time of release of the gas has to be limited or if there is significant longitudinal diffusion (Higgins, et al., 1958). Analysis of samples following pulse release, in which a known mass of gas is released, shows a rise and then a fall in concentration of tracer gas until it becomes too small to measure. In the steady release of tracer gas, where it is released at a uniform rate for a period of time, the subsequent analysis shows concentrations building to a constant value.

Tracer gas techniques have been utilized to determine leakages through sealed stoppings (Matta, et al., 1978), and to monitor ventilation flow and gas migration on longwall faces (Richards and Morris, 1978).

2.4 PRESSURE DETERMINATION

Pressure surveys are required to determine the pressure drops in mine airways which, with determination of quantity, allows the frictional resistance of roadways and the power absorbed to be calculated. The results are particularly applicable to the ventilation planning of any mine either in terms of changes in methods of working, rates of production, or to longer term development.

Pressure surveys may be carried out by measuring the actual static pressure drop between selected stations using by a U-tube pressure gauge and interconnecting tubing. Alternatively aneroid barometers may be used. Both methods are fully described in standard literature. Prior to 1969 European practice preferred the gauge and tube method, primarily due to the extensive calculations required for the corrections to the barometer survey results. Subsequently, McPherson (1969) provided a digital computer analysis to effect all the necessary corrections to the barometer survey results together with the provision of input data for ventilation network analyis if required.

It is now generally accepted that the gauge and tube method is the more accurate and more applicable to the accurate determination of pressure drops along a small number of airways. When an extensive survey is required, and overall accuracy of the complete traverse is more important than precision within individual airways, the barometer method may be preferred. The complete survey of a mine by the barometer is much less time consuming than the gauge and tube method.

Pressure surveys are particularly useful in evaluating the need for introduction of a booster fan, the precise location of the fan, and subsequent assessment of the effects of installation.

2.5 METHANE DETERMINATION

Methane surveys are required to determine sources and rates of methane emission in addition to quantitative and qualitative assessments of the environment in specific operative situations. Both mining conditions and mining method affect methane emissions in a mine. The development of highly mechanized mining methods has resulted in enhanced rates of methane emission. Merritts et al. (1963) noted that the introduction of continuous mining machines into room and pillar layouts both increased the gas emission volume and, due to the size of the machine, precluded the continuous flow of air across the face. Patching (1970) noted that additional degradation produced by machine mining could increase the emission rate. Although Joliffe (1970) found that emission rates on longwall faces did not increase pro rata with the production rate, nevertheless, the volume of gas did increase. Since there is a practical limit to the quantity of air which may be used to dilute methane emission, it may be necessary to introduce methods of methane drainage. This step must be preceded by an investigation to determine the sources of emission. Methane surveys form an integral part of these investigations.

The prediction of gas emission from longwall faces (Dunmore, 1981) is based on Airey's theoretical treatment of gas emission from broken coal and coal seams (Airey, 1968, 1971, 1979). The development of the MRDE method, against the background of methods used in other countries, was described by Dunmore (1979) and a detailed review of the approaches used in many countries was given by Curl (1978).

Application of the prediction theories requires specific data as to the sources of methane and peak emission rates which must be obtained from methane surveys. The use of continuous recordings from suitably positioned methane monitors is an advantage but must be augmented by surveys with portable methanometers.

The use of methane drainage techniques to reduce the quantity of methane reporting to the general body of ventilation has been the subject of a previous report (Dames & Moore/H.G. Stephenson, 1977a and b). However, even with methane drainage, Richards (1978) noted that methane surveys indicated that current drainage practice had little effect on the emission of methane at the coal face. Pickering (1969) identified four parallel corridors for airflow on a longwall face and showed that the concentration of methane varied across transverse sections of the face. The problems of gas flushing from the gob and of massive outbursts of methane, also require specific methane surveys to enable the ventilation to be brought under control.

2.6 ENVIRONMENTAL DETERMINATIONS

The extension of mining to greater depths, together with the effects of increased mechanization tend to create environmental problems due to dust, heat, and humidity which are influenced by the ventilation measures adopted.

2.6.1 Dust

Airborne respirable dust in the size range 1 to 5 microns is responsible for the development of pneumoconiosis or "black lung". Surveys are required to determine sources and concentrations of dust so that action may be taken to reduce the mean mass of respirable dust to safe limits. Concentrations are normally measured by means of a gravimetric dust sampler which removes from the air those particles normally deposited in the lungs by inhalation. Measurement locations are usually on the downstream side of coal faces, drivages, and transfer points. Goddard (1977) noted that airborne dust concentrations varied inversely with the ventilation quantity and recommended that velocities should not exceed 4 m/s if pick up of dust is to be avoided.

2.6.2 Heat and Humidity

The National Coal Board (1970) noted the desirability of measuring air temperature and humidity for two reasons:

a) to obtain a direct assessment of comfort
b) to determine the enthalpy and moisture content of the air in order to analyze heat and moisture pick up from the strata and machines with a view to provide suitable climatic conditions

Mucke and Voss (1971) considered that mine climate is the influence of air temperature, moisture, heat radiation, and air velocity on the well being of miners. One index of comfort level is the American Effective Temperature (ETA) which is based on miners stripped to the waist. The ETA is derived from a nomogram of dry and wet bulb temperatures and velocity of air. Whitaker (1981) considered that an ETA in excess of 28°C would impair a miner's ability to work effectively and safely. Verma (1980) concluded that an increase in the rate of air flow is a very effective way of improving mine climate resulting from both the increased velocity and the additional air to carry away the heat. The National Coal Board (1977) noted that the ETA will almost invariably be 1°C or more lower with homotropal than with antitropal ventilation.

3. APPLICATION OF COMPUTERS IN VENTILATION ENGINEERING

Technological developments have led to increasing use of computers in mine ventilation engineering. The earliest computer applications were devoted to the analysis of complex ventilation networks. Analysis was subsequently extended to include a wide range of parameters relating to the mine environment. Mathematical modelling of specific ventilation systems has evolved as a natural development. Routine analytical work may be handled by small desk-top computers. The introduction of microprocessors allows supervision, control, analysis, and display of the performance of ventilation systems, based on results from environmental monitors.

3.1 COMPUTER-AIDED VENTILATION NETWORK ANALYSIS

The solution of ventilation networks is essential in order to predict the effect on airflow of changes in the mine ventilation system. Such changes commonly include driving a new shaft or roadway, roadway enlargement, joining two roadways, introduction of a booster fan, relocation or addition of a main fan, and extension of the ventilation system as workings advance and new faces are commissioned.

In all but the simplest systems, the airways of a mine ventilation system constitute a network of combined series-parallel circuits with interconnecting roadways. An exact solution for simple series-parallel circuits may be arrived at algebraically. Complex networks, i.e. those which include overlapping and interconnected parallel circuits, cannot readily be solved algebraically, since both the direction and quantity of flow in interconnecting airways is unknown (Hartman, 1961). The mathematical solution of complex networks involves a method of successive approximation, developed by Hardy Cross in 1936, for general fluid network problems. The method is based on fundamental postulates of flow behavior, adapted from Kirchhoff's laws for electrical circuits. Application of this method to complex mine ventilation networks was introduced by Scott and Hinsley (1951-1952). The method applied by Scott and Hinsley assumes that air in mine roadways obeys Atkinson's Law, (pressure loss in an airway is proportional to the square of the air velocity) and assumes that the air is incompressible, i.e. no account is taken of changes in air density due to differences in temperature, pressure or humidity.

The Hardy Cross technique of successive approximation is accomplished by making an initial airflow distribution assumption for the network, that satisfies Kirchhoff's first law regarding continuity of flow. The law states that the algebraic sum of the currents (airflows) at any junction in a direct current circuit (ventilation network) is zero. By application of Kirchhoff's second law, which states that the algebraic sum of the voltage drops (pressure losses) around a closed circuit is zero, an approximate correction is computed and applied to each branch flow. The correction procedure is repeated until an acceptable degree of compliance with Kirchhoff's laws is obtained.

The solution of complex ventilation network problems by the method of successive approximation requires a substantial amount of computation and is, in most cases, unreasonably laborious and time consuming for manual calculation. In addition, when a variety of proposed changes to the ventilation system must be investigated, repeated trial solutions must be applied. The application of computers to aid the analysis of mine ventilation networks has been developed to obviate the need for excessively tedious, manual calculation. Computer-aided methods of network analysis are categorized into two distinct approaches: ventilation analogues and electronic digital computer programs.

3.1.1 Ventilation Analogues

Electrical analogue modelling of mine ventilation systems utilizes the principle that basic concepts of Kirchhoff's laws are applicable to both direct-current electrical circuits and fluid networks. A ventilation analogue consists of a network of electrical resistance elements assembled to identically model the pattern of airways which forms a mine's ventilation system. The current which flows through each resistance element represents the airflow through the corresponding airway, and the voltage drop across the resistance element represents the ventilation pressure difference between the ends of the airway. Whilst linear resistance elements, i.e. simple resistors which obey Ohm's law, were employed in the earliest analogue computers, refinements have since been made to provide a means of simulating the non-linear pressure/flow rate relationship of Atkinson's equation. As described by Williams (1979), the advent of low-cost microelectronic devices has made available electronic function generators which are being utilized in analogue computers as resistance elements to more precisely simulate the non-linear pressure/flow-rate characteristics of airways.

Since the electrical circuits which constitute the ventilation analogue must be physically connected in the specific manner which simulates the ventilation network, the analogue method has the significant disadvantage of inflexibility. This inherent disadvantage has been a primary reason for the analogue method's diminishing popularity in favor of electronic digital-computer methods. The principal future applications of the ventilation analogue are likely to involve permanent modelling of mine networks (an application which is relatively unencumbered by the analogue's inflexibility) and as an illustrative demonstration model to be used for training ventilation personnel (Williams, 1979).

3.1.2 Electronic Digital Computers

The ability of the electronic digital computer to perform repetitive calculations rapidly and inexpensively has been very successfully applied to the solution of complex ventilation network problems through the development of computer programs based on the Hardy Cross method of successive approximation. The immense data processing capacity and speed of the modern digital computer effectively render the once-limiting factors of network size and complexity relatively unimportant. This makes possible analysis of complex ventilation systems in far greater detail than was otherwise practicable. As was noted by McPherson (1969), this is achieved by diminishing the need to make simplifying assumptions regarding network configuration or airflow behavior.

Simplifying assumptions had to be made when using the analogue model because of the time and manpower needed, to manually reduce pressure survey data. With the digital computer, this constraint is largely eliminated. The digital computer's ability to perform calculations of airway resistance factors from mine barometer survey data, enables the ventilation engineer to input pressure data from large numbers of mine barometer survey observations, directly into the computer network analysis (McPherson, 1969).

In dealing with many mine ventilation systems, an analysis based on the assumption of incompressible flow, can give results of sufficient accuracy for most purposes. Computer programs which provide a more exact network analysis, by taking compressibility into consideration, are being applied to mine ventilation systems in which large differences in air temperatures and pressures occur. Both types of programs are described below.

a) Programs Based on Incompressible Flow

The more widely used computer programs developed for the analysis of mine ventilation networks are based on the assumptions of incompressible flow and compliance with Atkinson's law. They constitute remarkably flexible, large capacity, easy to apply tools to aid ventilation engineering. One such popular program developed by Rahim and Wang (1979), is applicable to any network whose configuration does not comprise in excess of 550 branches, 400 junctions, 350 meshes, or a total of 10 main and booster fans. The program is designed to handle natural splitting, fixed-airflow regulated branches, main and booster fans, leakages, and natural ventilation pressures. It computes the value of air flow in each branch, pressure loss at each junction of the network and operating point for each fan. For fixed-flow branches the program is capable of quantifying the pressure differences necessary across the branch to achieve the desired airflow.

In practice, the program requires that each branch be identified and that resistance of each branch be input. If airway resistance values are not available, as in the mine planning process, the program will generate these values if assumed friction factors and airway dimensions are input. Where required, the operating curve of pressure versus flow rate for any fan may be taken into account by providing input data comprising the coordinates of up to eight points on each fan curve. Natural ventilation pressures can be assigned to any branch and are treated as fans of constant pressure.

b) Programs Using Thermodynamic Analysis

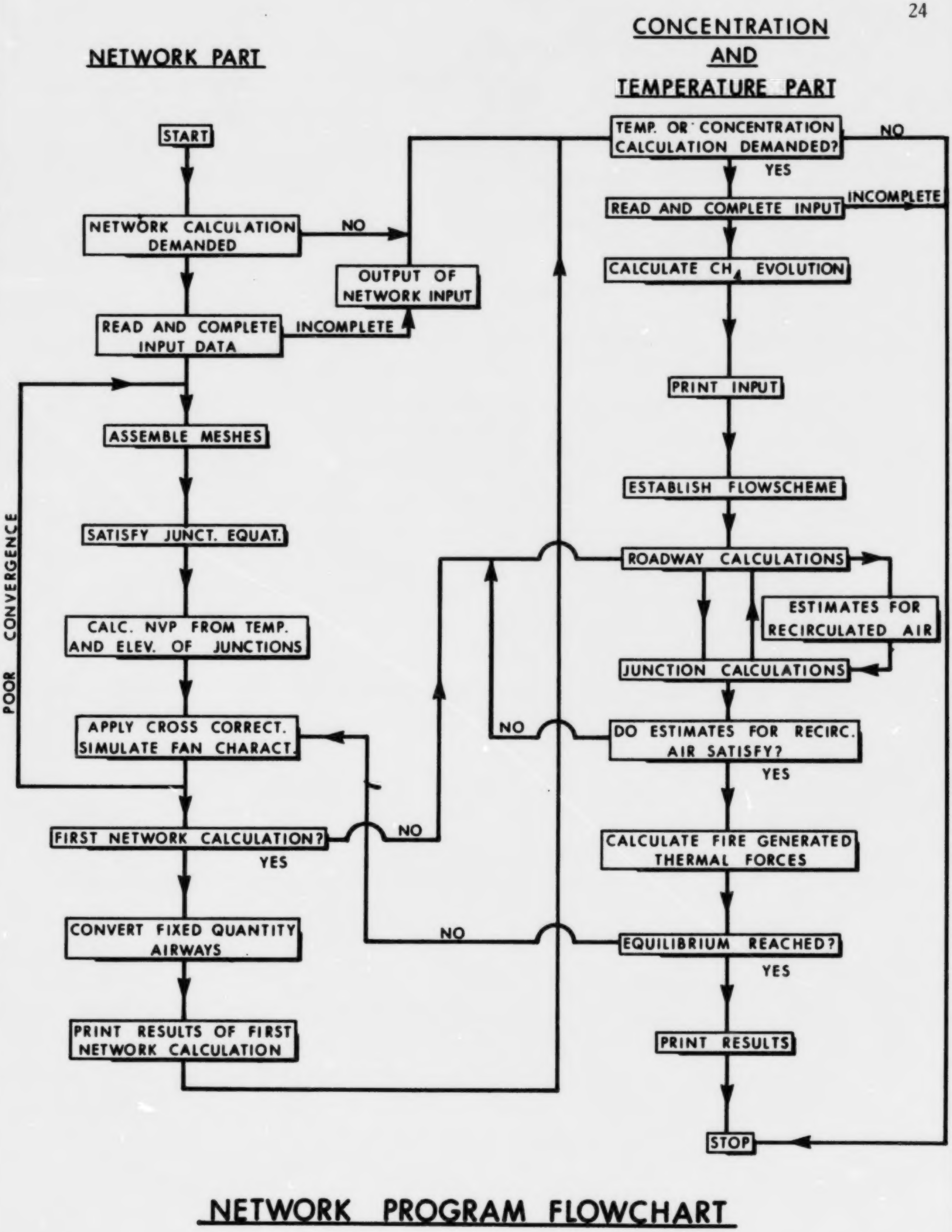
Mine ventilation network solution methods which assume incompressible airflow behavior can introduce considerable error when applied to ventilation systems in which large differences in air pressures, air temperatures, and/or airway elevations are encountered (Hall, 1981). A more exact method of solving such ventilation systems is possible using network solution methods which, rather than assuming incompressible flow, involves thermodynamic analysis of airflow behavior. This type of analysis is particularly valuable for simulating the effects an underground fire would have on the performance of a mine's ventilation system, and for simulating the environmental conditions in deep mines where climatic considerations must be addressed.

A digital-computer program that utilizes thermodynamic analysis to solve complex mine ventilation networks has been developed by Mr. Rudolph E. Greuer at the Michigan Technological University, under sponsorship by the U.S. Bureau of Mines. The program was developed to enable mine operators to simulate and quantitatively evaluate the effects a fire or other emergency would have on a ventilation system, and on alternative or projected configurations of the system. In addition to calculating flow for the mine network, the program provides for the consideration of other factors. These include methane emissions and concentrations; fire-generated contaminant liberations, concentrations and air temperatures. The program covers each airway and junction of the mine ventilation system (Edwards, 1981). This capability is essential for predicting the efficiency of fire detection systems and for identification of preferred escape and rescue routes in the event of a mine fire or other ventilation emergency.

The Michigan Technological University program employs a method of network analysis based on thermodynamic principles and gives proper consideration to temperature-induced ventilation forces (Greuer, 1979). The effect of a mine fire, fan stoppage, or other ventilation emergency on the performance of a mine's ventilation system, via airflow reductions and/or reversals, can be determined. In addition, airways in which excessive methane concentrations may result from disruptions of normal airflow are identified. This feature of the program is of obvious value for the analysis of the ventilation system where the methane concentration of the atmosphere, in any portion of the workings, is ordinarily being diluted to just meet the statutory limits. The program also permits the assessment of the resulting dangers from fire-induced airflow reversals, including the movement of fire-generated contaminants, such as smoke and carbon monoxide, into intake airways and other sections of the mine.

Practically every ventilation emergency imaginable may be simulated by the Michigan Technological University program (Greuer, 1979). A generalized flowchart (Greuer, 1979) of the steps with which the program performs a combined network, concentration, and temperature calculation is shown in Figure 3.1. Details of the program are documented in a program users manual prepared for the U.S. Bureau of Mines (Computer Sciences Corporation, 1980). A modification of the program has the ability to compute the real-time spread of contaminants throughout a mine network, rather than imposing a steady-state assumption (Edwards, 1981). Further alternatives to the program have been prepared to facilitate utilization by mine ventilation engineers (Edwards, 1982).

The output of the Michigan program consists of a listing of airflow rates and pressure losses in the airways and of methane concentrations, contaminant concentrations, and temperatures at each of the junctions. In addition, the airways in which airflow reversals occur and the airways in which critically low pressure drops develop are tabulated. The airways and junctions which have critically high


NETWORK PART
CONCENTRATION AND TEMPERATURE PART
START
NETWORK CALCULATION DEMANDED
NO
OUTPUT OF NETWORK INPUT
READ AND COMPLETE INPUT DATA
INCOMPLETE
ASSEMBLE MESHES
SATISFY JUNCT. EQUAT.
CALC. NVP FROM TEMP. AND ELEV. OF JUNCTIONS
APPLY CROSS CORRECT. SIMULATE FAN CHARACT.
POOR CONVERGENCE
FIRST NETWORK CALCULATION?
YES
NO
CONVERT FIXED QUANTITY AIRWAYS
PRINT RESULTS OF FIRST NETWORK CALCULATION
TEMP. OR CONCENTRATION CALCULATION DEMANDED?
NO
YES
READ AND COMPLETE INPUT
INCOMPLETE
CALCULATE CH$_4$ EVOLUTION
PRINT INPUT
ESTABLISH FLOWSCHEME
ROADWAY CALCULATIONS
ESTIMATES FOR RECIRCULATED AIR
JUNCTION CALCULATIONS
DO ESTIMATES FOR RECIRC. AIR SATISFY?
NO
YES
CALCULATE FIRE GENERATED THERMAL FORCES
EQUILIBRIUM REACHED?
NO
YES
PRINT RESULTS
STOP

NETWORK PROGRAM FLOWCHART

FIGURE 3.1

SOURCE: GREUER, 1979

methane concentrations, contaminant concentrations, or temperatures are identified, and the value of the critical parameter is listed.

3.2 MATHEMATICAL MODELLING

The use of main computers to mathematically model the mine climate is reported by Voss (1969), McPherson (1974) and Whittaker et al. (1977). The mine is sectionalized and input data includes depth of workings, virgin rock temperature, heat and humidity coefficients, production rate, power consumption and extent of air conditioning. Morris and Walker (1982) report the modelling of booster fan controlled recirculation systems; determination of the relative cost benefits of alternative solutions to ventilation problems (such as booster fan installation or roadway enlargement); and the application of combined surface and underground methane extraction systems. Cline (1981), in a publication of the Colorado School of Mines, lists details of modelling programs addressing estimation of diesel exhaust contamination (Stefanko et al., 1972) and simulation of methane and water flow in coal (Price and Abdulla, 1972).

The model described by Ramani and Owili-Eger (1973) is developed to determine methane emission rates for use in planning a methane drainage system and predicting the performance of a proposed ventilation network. Both steady state and unsteady state gas flows are included, and have been modelled as second order, non-linear, partial differential equations. Finite difference equations are used to obtain numerical solutions to the models. Dunmore (1981) reports on current methane prediction techniques being developed and tested at the National Coal Board's Mining Research and Development Establishment. These models are based on Airey's theoretical treatments of methane emission from coal seams and broken coal (Airey, 1968, 1971, 1979) and are described in Section 4.1 of this report. Computer models for the analysis of methane extraction systems are available to forecast system performance under various operating conditions of layout, pipeline capacity, suction pressure, ambient conditions, drained gas purity, gas flow rate and exhauster characteristics (Morris and Walker, 1982).

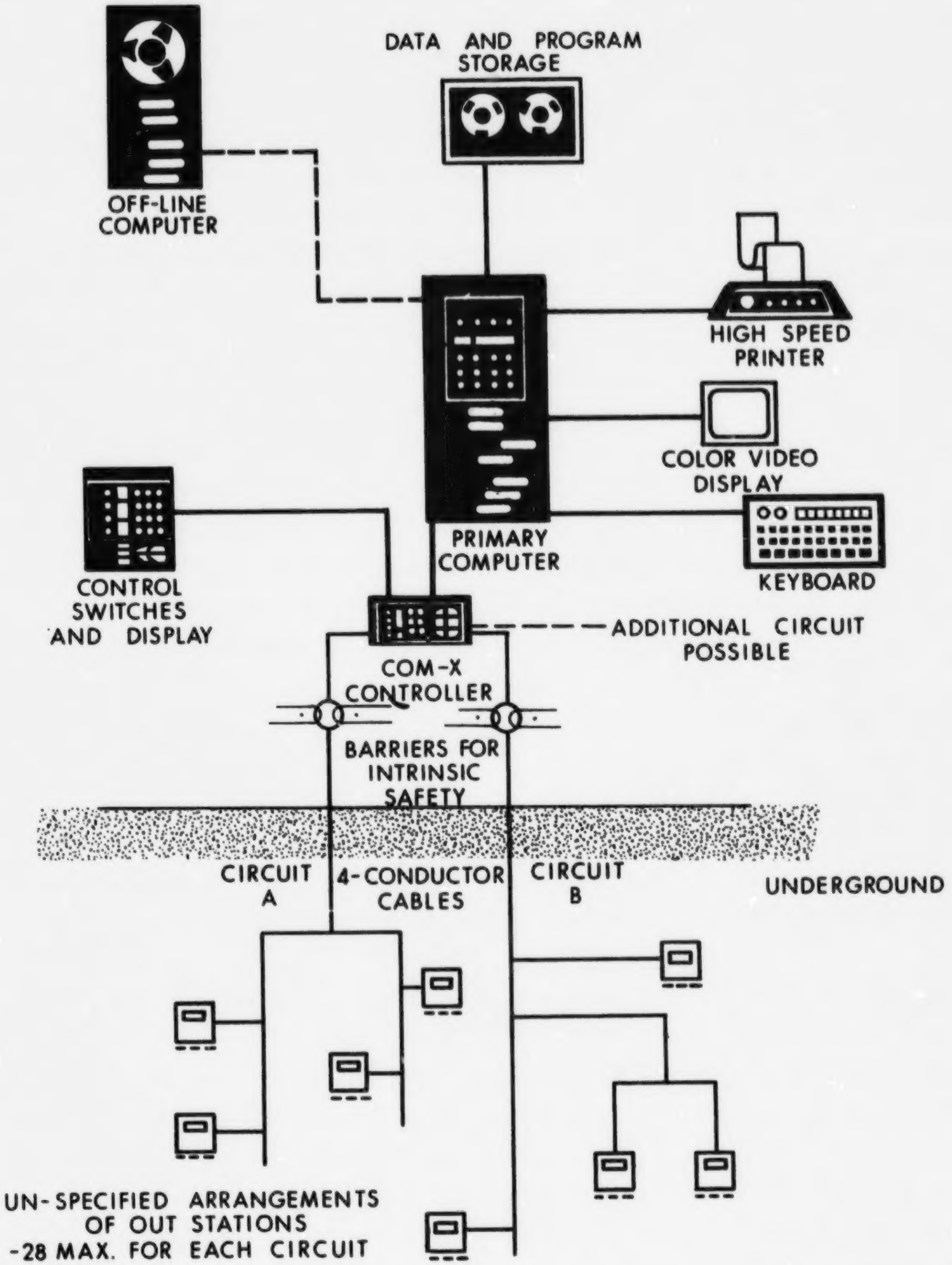
The manner in which mathematical modelling can be used to compare alternative ventilation strategies to secure effective ventilation is described by Highton (1982). Twelve different strategies to ventilate the future workings of McIntyre Mines Ltd. No. 5 Mine were contrasted to enable choice of the optimum design.

Small desk-top computers are being used for routine analytical work associated with evaluating the effects of limited ventilation changes, auxiliary ventilation, methane emission, methane drainage, air conditioning and general forecasting (Morris, 1979). Such equipment is adaptable, inexpensive, interactive, and programmable by mine ventilation engineers to enable forecasts to be made of practical situations.

3.3. DATA PROCESSING

The typical role of computers in remote ventilation monitoring and control systems is illustrated in Figure 3.2. The front-end computer, or communication controller, coordinates communications with underground outstations, continually updating its data registers with status information every few seconds. The primary computer processes the current information received by the communication controller and presents the data in summary form to the operator for review. The operator's keyboard commands are interpreted by the primary computer and relayed to the controller for transmission underground. In addition, the primary computer's functions may include control of a color video monitor and high speed printer, interface with the magnetic or paper tape program storage medium, drive of remote monitoring sites, and communication with auxiliary computers.

Microcomputers are also used to drive tube bundle systems for routine monitoring, control and display (see Section 3.2). Microprocessors are employed for the reduction of data to standard conditions and single values, (such as the CO/O_2 deficiency ratio) drained pure methane flow rate, effective temperatures and the automatic identification of parameter values which exceed a pre-determined limit or which change at a significant rate.


DATA AND PROGRAM STORAGE
OFF-LINE COMPUTER
HIGH SPEED PRINTER
COLOR VIDEO DISPLAY
PRIMARY COMPUTER
KEYBOARD
CONTROL SWITCHES AND DISPLAY
ADDITIONAL CIRCUIT POSSIBLE
COM-X CONTROLLER
BARRIERS FOR INTRINSIC SAFETY
CIRCUIT A
4-CONDUCTOR CABLES
CIRCUIT B
UNDERGROUND
UN-SPECIFIED ARRANGEMENTS OF OUT STATIONS -28 MAX. FOR EACH CIRCUIT

REMOTE MONITORING AND CONTROL

SYSTEM DIAGRAM

FIGURE 3.2

NorWest Resource Consultants Ltd

SOURCE: MINETICS CORPORATION, 1981

Further applications of microprocessor-aided systems relate to continuous supervision of main, booster, and auxiliary fans; surface methane extraction and utilization plants; early warnings of abnormal environmental situations (Morris and Walker, 1982), control and "fail-safe" functions in methane drainage (Morris and Gray, 1977), controlled booster fan recirculation systems, and environmental control. Intensified application of environmental monitors, may provide data in amounts that exceed the capacity of existing data transmission systems. The use of microprocessors in outstations would allow processing prior to transmission and would reduce the amount of data to be handled.

4. MINE FANS

4.1 MAIN FANS

The ventilation of underground mines is normally provided by the surface installation of fans which may either exhaust air from the mine or force air into it. Exceptionally, some large mines are ventilated by "push-pull" systems which utilize both methods. The basic types of fans are the axial flow, in which the air leaves the rotor in a direction substantially parallel to the axis, and the radial flow, in which the air leaves the impeller in a direction substantially at right angles to its axis.

Choice of fan type will depend on a number of factors and it is a prerequisite that ventilation planning should determine air volume flow rate requirements and network resistances likely to be encountered over the mine life. In general terms it may be stated that axial flow fans are favoured where a high flow rate/pressure ratio applies and radial flow fans have advantages for a low flow rate/pressure ratio.

4.1.1 Efficiency

Both types of fans have operating static efficiencies in excess of 80 percent. The introduction of the aerofoil blade section in the backward bladed radial flow fan substantially improved its efficiency and some manufacturers claim maximum static efficiencies of about 86 percent.

4.1.2 Performance

The performance of the axial flow fan can be varied to meet a wide range of mine resistances at high operating efficiency but the fan characteristics provide for high flow rates at relatively low pressure. Radial flow fans are less flexible in operation but provide a broader operating range at peak efficiency and are more suitable where high ventilating pressures are required. In recent years, the development of larger mines with geographically concentrated production faces has required substantial increases in surface fan output up to 7 kPa and 450 m^3/s with installed powers of some 3000 kW

(Morris and Walker, 1982). As a result, increasing use has been made of the radial flow fan.

4.1.3 Variation in Performance

The axial flow fan can be varied in duty by:

a) change in speed
b) blade pitch variation, which can be implemented whilst the fan is in motion on an auto variable pitch rotor blade machine
c) partial deblading
d) fitting additional stages
e) any combination of the above

The radial flow fan can be varied in duty by:

a) use of variable inlet guide vanes
b) change in speed
c) use of extensible blade tips
d) any combination of the above

4.1.4 Stalling

The axial flow fan is subject to stalling as the air velocity through it decreases and the duty point of the fan nears the top of the characteristic curve. At this point, fan operation becomes unstable and minor changes in mine resistance result in a breakdown of airflow conditions through the fan. The radial flow fan has no noticeable stalling point and is better suited to accommodate fluctuation in mine resistance.

4.1.5 Power Characteristics

Both types of fans have non-overloading characteristics. The axial flow fan absorbs maximum power at approximately the point of maximum efficiency over the entire range of blade pitch adjustment and the power decreases as the volume flow rate increases towards the condition of free delivery. The radial flow fan absorbs maximum power at a volume flow rate greater than that at which peak efficiency occurs; power then decreases as the flow rate increases towards the condition of free delivery.

4.1.6 Reversibility

Reversal of direction of flow through the axial flow fan is easily obtained by reversing the direction of rotation of the rotor. Under these conditions the fan will operate inefficiently and at a considerably reduced level of performance. The ability of the fan to effectively produce flow reversal in a mine by this means is dependent on the local mine ventilation geometry and magnitude of natural ventilation. Airflow reversal in a mine ventilated by a radial flow fan is effected by operation of an arrangement of doors which are an integral part of the fan layout. After airflow reversal, the fan continues to rotate in the normal direction, so that a fan which normally exhausts air from the mine, forces air into it.

4.1.7 Space

The radial fan takes up considerably less ground space than the equivalent axial flow fan for the same duty.

4.1.8 Cost

The basic cost of the mechanical parts of a radial flow fan are usually less than the costs of an axial flow fan for the same duty. No general comparison can be made of the installed costs since this is primarily a function of the amount of civil engineering and building work required.

4.2 BOOSTER FANS

The term "booster" is applied to fans of both axial and radial flow types installed underground when they are required to handle the whole of the air present at the fan location. The booster fan has a specific application in sub-marine mines where additional inbye access from the surface is not possible and mine resistances are large due to extensive development offshore. Booster fans have been used extensively at 26 Colliery in Cape Breton (Section 8.4). The installation of a booster may be more economical than the provision of a larger surface fan which may create intolerably high pressure differences in outbye parts of the mine.

Local installation of a booster fan has the objective of either increasing the available airflow in that part of the mine or of minimizing leakage by reducing pressure differences. The alternatives to the introduction of booster fans are reductions of ventilation resistance by roadway enlargement or an increase in the output of the main fan. If additional airflow is required in only one section of a mine then speeding up the main fan may result in increased leakage, the need to introduce increased regulation in low resistance splits, and greatly increased power consumption. In 1971, the Institution of Mining Engineers considered that, in general, the introduction of a booster fan was more economical provided the period of usage did not exceed 4 years. For lengths of time greater than this, roadway enlargement was the most economical solution.

Tabor (1969) considered that the following advantages accrued to the introduction of booster fans:

a) Initially it is less expensive to install a booster fan than to drive or enlarge an airway, but the duration of use (and accumulated running costs) need careful consideration.
b) Booster fans may be used to avoid high pressure differences at the surface, shaft bottom, and across regulated districts. Their use may reduce ventilation power costs.
c) Installation costs can sometimes be less than for a replacement surface fan.
d) Booster fans may be used to minimize pressure differences throughout in-the-seam workings, especially in areas liable to spontaneous combustion.

Statham (1943) considered that booster fans were primarily employed underground for the following reasons:

a) To increase the airflow passing in a single split of high resistance
b) To increase the airflow passing through a number of splits
c) To reduce the leakage between intakes and returns by reducing the pressure which would be necessary if the surface fan was speeded up to increase the airflow
d) To reduce the pressure required and the tendency to leakage, in mines liable to spontaneous combustion

4.2.1 Siting

The siting of booster fans is of particular importance if the required results are to be achieved. The essential prerequisite prior to installation is the completion of an airflow and pressure ventilation survey and determination of the resistances of the various ventilation circuits which may be affected. Richardson (1974) noted that it was relatively easy, with the aid of a digital computer network program, to derive the accurate fan duty and position in the network which would give the required air distribution. This view is probably over-simplified since due consideration must be given to the variations in methane concentration at the return airway locations under consideration and also to the problems of critical pressure and recirculation.

Critical Pressure: The critical pressure of a booster fan is that value at which its operation will cause airflow to cease in outbye ventilation splits of relatively lower resistance. Operation of the fan in excess of the critical pressure may result in air reversal in outbye splits. Since the critical pressure depends on the relative resistances of the inbye splits and the main airways, the need for these to be determined is paramount. If the resistance of the main airways is such as to absorb a large proportion of the pressure produced by the surface fan, then the pressure which can be applied by the booster fan before causing air reversal is limited. In this circumstance the booster may require to be located in a position to benefit more than one split and, hence, splits of low resistance will have to be regulated.

Recirculation. The effect of a booster fan is to cause increased pressure on the discharge side and reduced pressure on the suction side of the fan. The change in pressure distribution may result in increased leakage of air in strata surrounding the booster fan installation, thereby causing recirculation, resulting in loss of volumetric efficiency and creating potential hazards due to methane content of the recirculated air. Statham (1943) considered that the danger of recirculation increased in direct proportion to the distance inbye in a single split in which a booster fan was installed. He concluded that the fan should be installed as near the outbye end of the split as possible, and subject to due consideration of the possible leakage paths such that the fan is located outbye of such paths when any leakage will tend to flow from intake to return.

The Institution of Mining Engineers (1971) noted that leakages, (even inbye), where they result from relatively large pressure differences between adjacent airways may be nullified by carefully sited booster fans. The example is cited of the inadequate ventilation of a face due to excessive leakage outbye at two overcasts and two sets of doors located in geologically weak ground. The installation of an 0.8 m diameter booster fan inbye of the leakage sites neutralized the pressure differences and resulted in more effective use of the available airflow. The Report of the Commission of Inquiry into the explosion in No. 26 Colliery, Nova Scotia (Elfstrom, 1980) included consideration of a booster fan installation in the intake airway of a longwall district to minimize gob leakage and increase face ventilation as a result of the reduced pressure difference between intake and return. In this situation localized leakage at the fan site was to be minimized by the provision of an airlock and extensive grout injection of the surrounding strata. The principle of using a booster fan to control leakage between intake and return is also of particular benefit in the prevention and control of spontaneous combustion.

4.2.2 Types of Booster Fan

In "A Review of Auxiliary and Booster Fan Ventilation Practice in Coal Mines" (Health and Safety Executive, 1978) it was considered that there were two categories of booster fans:

a) the standard unit generally similar but larger in size and duty than an auxiliary ventilation fan, which may be installed alone or in series/parallel clusters when higher duties are required and

b) the specially designed fan built specifically for a particular duty and site

In relation to these two categories it was considered that the advantages accruing to each were:

Cluster Type

a) the ease with which standby ventilation facilities can be provided

b) the lower capital cost of the installation

c) the flexibility and range of duties which can be readily provided to meet changing ventilation requirements

Purpose Built Type

a) the higher efficiencies and lower running costs which can be expected

b) the reduction in the amount of electrical and mechanical equipment which needs to be provided

c) the simpler monitoring arrangements which can be provided, particularly with regard to vibration detection and temperature sensing

The report concluded that selection of either purpose built or cluster type fans would depend on site specific criteria. In this regard Richardson (1974), in considering the application of underground booster fans to collieries in the Staffordshire Area of the National Coal Board, selected cluster type fans of the axial flow type in preference to radial flow or purpose built fans of either types for several reasons:

a) the short term installation may render the purpose built fan difficult to reuse if the duty is site specific

b) the range of duty with cluster type fans was more flexible

c) standardization, in terms of fan type; and installations, was more easily attained.

Currently, wider use is being made of booster fans with total installed power up to about 1000 kW. Typical outputs range from 35 m^3/s at 2 kPa to 100 m^3/s at 5 kPa from clusters of 1.22 m diameter, 55 kW conventional axial flow fans with efficiencies of the order of 60-70 percent (Morris and Walker, 1982). Static efficiencies of 75-85 percent are available from custom designed radial and axial flow fans.

Although uncontrolled recirculation is a hazard with all underground fans, booster fan controlled recirculation methods are reported by Morris and Walker, (1982) to have advantages in specific situations. It is possible to demonstrate that such a system may be used to increase face airflow rates and velocities, and decrease airborne dust concentrations. Methane concentrations would increase but only to a level determined by dividing the total emission by the incoming fresh air volume flow rate. The application of monitoring is an essential prerequisite if proper control is to be obtained over a technique which has appreciable inherent risk.

4.2.3 Monitoring

The development of remote monitoring for underground booster fans was described by Wastell and Robson (1976). They reported that the system was reliable and safe and had resulted in booster fans being allowed to run unattended for periods of 8 hours when men were in the mine. The systems have since developed and it is reported (Health and Safety Executive, 1978; Schwartz, 1981) that the following parameters are currently being monitored: fan running, vibration, motor casing temperature, motor bearing temperature, motor overload, smoke, methane, air velocity and pressure and scanning by television.

It has been recommended that, when inbye workings are heavily dependent upon the ventilation provided by a booster fan, the methane concentration and airflow should be continuously monitored.

4.2.4 Safety

In relation to the safety aspects of underground booster fan installations, the recommendation of the UK National Committee in "A Review of Auxiliary and Booster Fan Ventilation Practice in Coal Mines" (Health and Safety Executive, 1978) are considered to be generally applicable and are stated as follows:

a) Booster fans of the purpose built type should be designed to a specification similar to that used for main fans at the surface.
b) Manager's rules should be established for the installation, maintenance and operation of all booster fans.
c) In all mines where booster fans are installed, ventilation survey information should be updated periodically and when it is deemed necessary as a result of changes in the ventilation system.
d) The manager should be advised immediately of any information resulting from updated surveys or computed conditions, including the effects of stoppages of fans which could constitute a hazard.
e) The use of vibration monitoring is recommended on all booster fans.

4.3 AUXILIARY FANS

4.3.1 Fan Types

An auxiliary fan is a fan used below ground wholly or mainly for the ventilation of a heading, drift or blind end and which handles, by means of ducting, only a proportion of the airflow in the roadway in which it is sited.

There are two main types of auxiliary fans, i.e. axial flow and radial flow. They are available in the following configurations:

a) The in-line radial flow fan, in which the air enters the eye of the impeller and is turned through 90 degrees within the impeller. The air is then turned through another 90 degrees to flow along the bifurcated fan casing. The motor is mounted in a pod out of the airstream.

b) The in-line axial flow fan, in which the motor is mounted within the airstream behind the impeller.

c) The bifurcated axial flow fan, in which the motor is mounted outside the airstream within a pod and the air flows along the bifurcated trunking of the casing.

The "Review of Auxiliary and Booster Fan Ventilation Practice in Mines" (Health and Safety Executive, 1978) considered the merits of the alternative auxiliary fan configurations as follows:

In-Line Axial Flow Fans

The main disadvantages of in-line axial flow fans for the majority of ventilation systems are:

a) The motor can be exposed to dust and water

b) It has limited pressure development and, if designed for high performance, the characteristic could have a severe stall condition

c) High noise level

d) The design is more suited to forcing ventilation systems

Bifurcated Axial Flow Fans:

The main advantage of the bifurcated design is that the induction motor is mounted out of the airstream and therefore should not be contaminated by dust or water. Disadvantages include:

a) modest pressure development

b) high noise level

c) hollow aerofoil blades can have a limited life when used with dust collectors

d) there are at present undesirable features related to the method of motor mounting

In-Line Radial Flow Fans

The in-line radial flow fan became widely used because the design included the advantages of the bifurcated type of machine together with a performance characteristic giving considerably higher pressure development. Other advantages are that the smaller diameter impellers give lower blade tip speeds, and better entry conditions are achieved by the inlet cone; the noise levels are lower than those of axial flow fans of the same power.

Disadvantages of the design are:

a) examination of the induction motor and fittings is difficult because of poor accessibility
b) hollow steel blades can wear quickly and hold water when used with wet dust filtration units
c) the cable entry arrangements are particularly susceptible to mechanical damage

4.3.2 Fan Combinations

Auxiliary fans can be run singly or together in various combinations to achieve specific objectives. The basic methods are:

a) single fan
b) fans in series
c) fans in parallel
d) fans in a series/parallel combination

In considering these alternatives, Barker and Highton (1981) concluded as follows:

Single Fan

After determination of the type of ducting to be used, and after consideration of fan characteristic curves, the choice of a single fan capable of generating the required air presssure and flow produces the simplest ventilation system.

Fans in Series

When two fans are coupled in series, the airflow passing through each is the same and if the fans are identical, they will develop, theoretically, twice the pressure of one fan, i.e. the presures developed are added. If the fans are not identical, the net effect upon the airflow is dependent on duct characteristics.

Fans in series may be directly coupled to each other on the outside of the heading or may be spaced at intervals along the duct. In the latter case, uncontrolled operation may lead to duct leakage and recirculation. Preferably, all fans should be sited outside the heading to be ventilated.

Generally, if the flow rate delivered at the face of a heading is too low, there is an instinctive desire to add another fan in series. However, to achieve optimum results, the system resistance should be used in association with the series and parallel compounded fan characteristics to select the best combination.

Fans in Parallel

If fans with incompatible characteristics are arranged in parallel, unstable operation may result, with backflow through the impellers and oscillation of the delivery column. To ensure that operating conditions are stable it is necessary that fans have identical operating characteristics and that the range of fan duty in relation to the combined characteristic permits stability.

4.3.3 Ducting

The limitation on the amount of air which can be circulated to the face of a heading is usually imposed by the type of ducting in use, its diameter, and particularly the standards of installation and maintenance of the ducting. In British coal mines for example, three standard types of ducting are currently in use. These comprise rigid steel, flexible and spirally reinforced flexible ducting. Flexible ducting of the non-reinforced type can only be used where the auxiliary ventilation system is forcing. Of these three types of ducting the resistance to airflow is lowest with steel and highest with the spirally reinforced ducting (Health and Safety Executive, 1978). In Canada rigid fibre glass ducting may be used in place of rigid steel providing it meets the requirements for anti-static properties.

The three parameters which affect delivery of air through ducting are the leakage coefficient, pressure loss and correction factor. These parameters were defined by Barker and Highton (1981) as:

a) Leakage coefficient is the airflow which would leak from a unit length of ducting under uniform unit pressure
b) Pressure loss related to leakless ducting and the leakage coefficient are obtained from standard graphs (National Coal Board Bull. 60/220)
c) Correction factor is the ratio of airflow at the fan to the delivered airflow

The development of longer, larger, rapidly advancing headings, has led to continuing consideration of ducting size, shape and type. Retangular ducts have the benefit of space economy (Hagood, 1979). Rapid and continuous extension are prime requisites for any type of ducting. Various types of duct couplings have been designed for ease and security of joint connection.

4.3.4 Auxiliary Fan Selection

Fan selection is determined on the basis of site specific data including airflow to be delivered, methane emission rate, airflow available in the roadway, type, diameter and length of ducting to be used.

Current auxiliary ventilation practice uses larger fan sizes (up to 90 kW) larger duct diameters (up to 1.4 m), higher flow rates and pressures, and more exhaust systems in order to control dust. Morris and Walker (1982) report a substantial expansion in the use of 37 kW in-line radial flow fans handling 10 m^3/s at 5 kPa and efficiencies up to 65 percent in the United Kingdom.

4.3.5 Monitoring

In the "Review of Auxiliary and Booster Fan Ventilation Practice in Mines" (Health and Safety Executive, 1978) it was considered that the most important aspects to monitor are those which establish that the system under consideration is performing the function for which is was designed, i.e. that the airflow at the face is above the minimum statutorily required and that its quality is satisfactory in terms of methane dilution. It is recommended that any system should include monitoring of air quality and quantity as a minimum objective. Sensors are now available which will indicate when predetermined levels of these two parameters are reached. There does not appear to be any need to indicate actual values and indication of a rise of methane concentration to a pre-determined value or of a fall in air flow to a predetermined level is considered sufficient for a simple monitoring system.

Moses and Jackson (1979) reported on the introduction of auxiliary fan monitoring at Peckfield Colliery where the fan was situated some 5 miles from the downcast shaft and the data were transmitted over the conveyor control systems. The objective was to transmit the following data to the surface.

a) The concentration of methane in the fan duct; the fan was an exhausting installation and an alarm had been set at 1 percent methane.
b) The air flow in the duct; an alarm had been set to determine when the airflow had fallen 10 percent below the stipulated minimum.

The equipment consisted of a short steel tube, of the same diameter as the duct, inserted into the duct at a position not more than 1 week's advance from the face of the heading. The steel tube housed a sensing head associated with a BM 1 transducer to monitor methane and an anemometer associated with a BA 2 transducer to monitor airflow. The electrical output from these sources was fed into the telemetering cable used for the conveyor control and the information was displayed on the surface control desk by a flashing light and an audible warning. There was no intention to remotely control the fan in any way and on the activation of the warning signals it was necessary to alert underground personnel to investigate.

5. VENTILATION STRATEGIES

5.1 DANGERS ARISING FROM METHANE AND DUST

The most serious incidents in underground coal mining, in terms of loss of life and damage, are usually explosions and fires. Most explosions are initiated by the ignition of methane/air mixtures though coal dust is often a factor in propagating the original explosion. Generally speaking the greatest loss of life in these circumstances is caused by the gaseous products of combustion, though both blast damage and fire damage may also be severe.

The dilution and/or removal of methane from the mine by ventilation and drainage systems therefore becomes a primary concern of the mining engineer at both a planning and operating level. This concern, while more critical in situations where emission rates are very high, also exists where methane is rarely encountered since, in the latter case, explosive concentrations of methane may go undetected due to complacency or inexperience. Extremely low methane emission rates are encountered in the Alberta Plains coals and in some areas of Nova Scotia.

The danger from methane is not confined to its inflammable nature. In certain areas, particularly in the Canadian Rocky Mountains, coal seams liable to outbursts (very rapid emissions of large quantities of methane and fine coal,) are encountered. The danger from asphyxiation must then be considered.

In extreme circumstances, where ventilation circuits are extensive and emission rates are very high, the efficiency of the ventilation system, in reducing the methane content of the air in the workings, may determine whether the mine operation is technically and economically feasible.

The danger from coal dust arises from its explosibility, inflammability and the physiological effects on mine workers. Mechanization has resulted in a serious increase in the percentage of fine coal produced underground. Concurrently, the dangers of fine coal in the airstream have been more accurately identified and recognized. The role of improved ventilation systems in diluting dust to acceptable levels is important, though eradication or suppression of the dust at source remains the primary goal of the mine safety engineer.

5.1.1 Sources of Methane

A thorough discussion of sources of methane is included in "A Study of Methane Drainage Practices in Coal Mines and their Potential Application to Canadian Conditions" (Dames & Moore, and H.G. Stephenson Mining Consultants, 1977 pages 15-29).

The source and emission patterns of methane depend to a large extent on the method of mining. For example, in room and pillar workings where gobs are not extensive, the primary source of methane will be the seam being extracted. On the other hand, where extensive gobs are created by longwall operations, methane sources include the seam being worked, adjacent strata and coal seams, long conveyor lines, broken coal, and the gobs themselves.

In a typical longwall district, methane originates from three main sources:

a) the coal seam itself, by direct and immediate desorption from coal broken by the mining machinery, and from the newly exposed face;
b) adjacent coal seams within the influence of the seam being worked; and
c) associated strata, particularly sandstones which may act as gas reservoirs.

Large quantities of methane may accumulate in the longwall gobs behind the face support lines. Rapid barometric changes or heavy gob caving can flush the gas out into the face area and interrupt mining operations.

The volume of methane produced from broken coal varies with the type of coal and, to some extent, the sizing of the material. Particle sizing affects the rate of desorption because with smaller particles, the surface area is larger per unit weight of coal.

In any mining section it can be assumed that the quantity of methane released will increase with the rate of production. If the desorption rate of a seam is high, then most of the methane released will report to the main returns relatively quickly. However, if the desorption rate is low, emission can continue for much longer periods after coal production has ceased. This continuing desorption derives from prolonged emissions from coal ribs, remnant coal in gobs and from the face itself.

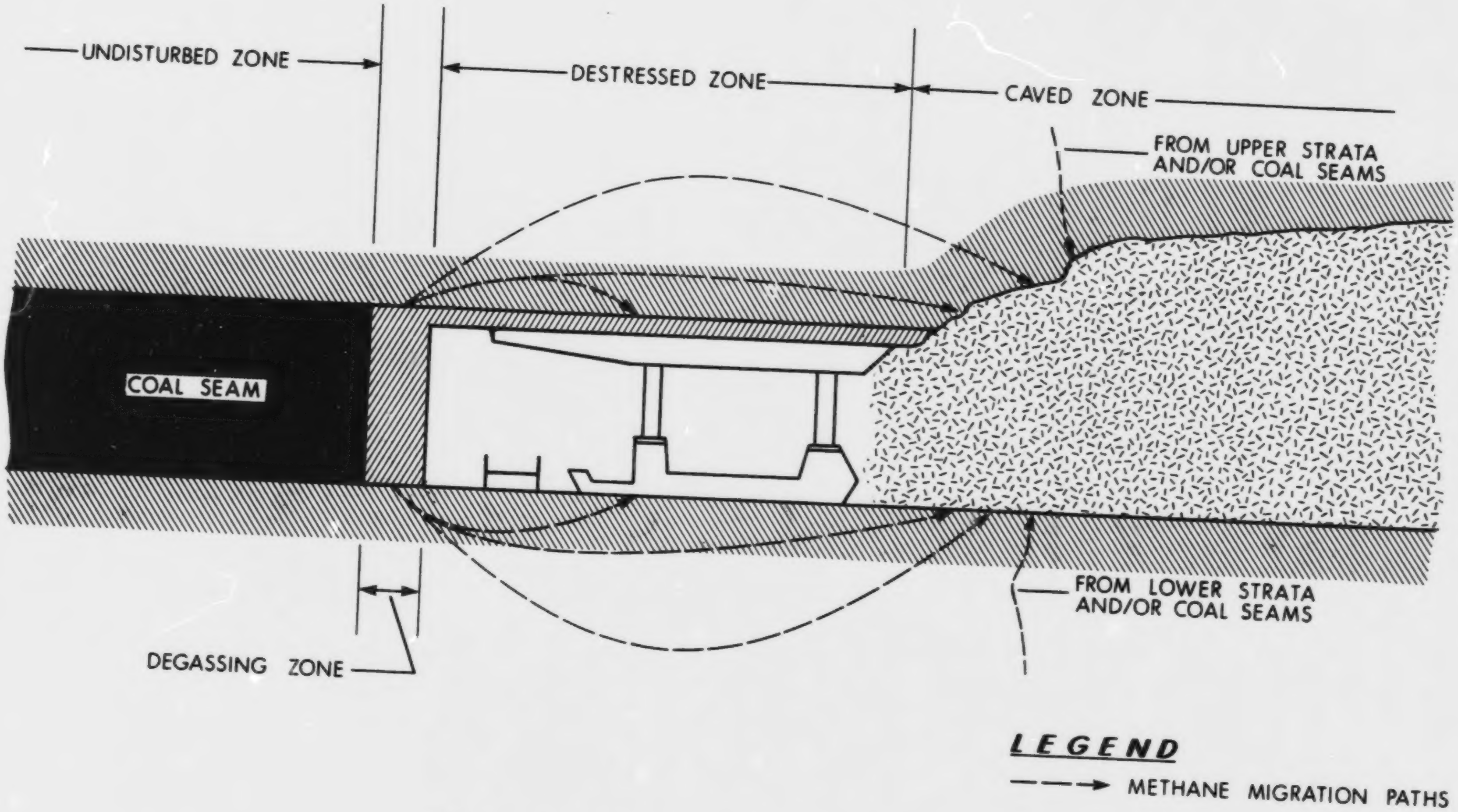
Figure 5.1 is taken from the report of the Commission of Inquiry into the 1979 explosion at No. 26 Colliery, Glace Bay, Nova Scotia (Elfstrom, 1980). This figure shows paths of methane migration from the zone in advance of the coal face. The gas flows into the gob through zones of bed separation and fracturing in the immediate roof and floor strata. The ability of the longwall supports to prevent bed separation will directly affect gas migration, therefore good strata control is a critical factor in reducing and controlling emission of methane.

5.1.2 Prediction of Methane Emission

Ventilation requirements are largely dependent upon the rate of methane emission into the mine workings, so determination of projected mine ventilation requirements must involve prediction of methane emission rates. Methods of prediction in the past were dependent mainly upon experience with methane emission rates in neighbouring workings. The development of mines in previously unworked areas brought about new methods based on solving functions that relate to specific mining and geologic factors, using data gathered during detailed pre-mining investigation.

European and Russian methods of predicting methane emissions all involve consideration of certain common parameters (Curl, 1978):

a) The stratigraphy above and below the worked seam
b) The desorbable gas content of the worked coal seam, and of adjacent seams and strata
c) The zone of gas emission in both the roof and floor strata
d) The degree of gas emission from adjacent seams and strata


UNDISTURBED ZONE
DESTRESSED ZONE
CAVED ZONE
FROM UPPER STRATA AND/OR COAL SEAMS
COAL SEAM
FROM LOWER STRATA AND/OR COAL SEAMS
DEGASSING ZONE
LEGEND
METHANE MIGRATION PATHS

DIAGRAMATIC REPRESENTATION OF METHANE MIGRATION ROUTES

FIGURE 5.1

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Resource Consultants Ltd

SOURCE: ELFSTROM 1980

Curl defines the zone of gas emission as being the three-dimensional block surrounding the workings, from which gas is released and flows into the mine workings or the methane drainage system as a result of mining. The "degree of gas emission" is the proportion of gas from a particular source which will enter the workings during the life of the district. Prediction methods differ in their definition of the zone of gas emission and the assumed variation of the degree of gas emission within the zone (Curl, 1978).

The emphasis in the USA has been on the prediction of methane flow into room and pillar workings. Because strata relaxation adjacent to room and pillar workings is relativey minor, the methods have focused on modelling the methane flow through the worked seam. In comparison, European and Russian methods account for the emission from adjacent strata as well as from the worked seam in longwall operations (Curl, 1978).

The basic methane content of a coal seam, i.e. the total volume of the methane sorbed per tonne of coal, together with the methane content of other seams disturbed by the workings of a coal face, is a major factor controlling methane emission rates (Jollife, 1970). There are two general methods of measuring the desorbable gas content of coal seams: direct and indirect. In the direct methods, coal samples are taken from the seam and pulverised within a sealed mill to release the inherent gas content. Suitable corrections are made for gas loss prior to sealing the sample and for the quantity of gas which remains sorbed by the coal at the final mill pressure. The indirect methods involve measurement of seam gas pressure in situ, compared with laboratory-measured gas content/pressure relationships. The determinations are complicated by the hydraulic effect of water in the strata and by the fact that the gas content/pressure relationship must be determined at the same temperature and moisture content as prevails in the seam, because small changes in moisture content have a marked effect on the methane sorption capacity of the coal (Curl, 1978; Dunmore, 1979).

A method for prediction of methane emission from longwall faces is being developed and tested at the Mining Research and Development Establishment (MRDE) of the National Coal Board (Dunmore, 1981). Airey's theoretical models of the behaviour of methane emission from broken coal, worked coal seams, and from seams adjacent to the worked seam (Airey, 1968, 1971, 1979) are used to provide estimates of the rate of methane emission into the airways and the potential rate of capture by cross-measure methane drainage boreholes. Emission rates are calculated from knowledge of the depth of the working, the sequence of coal seams in the disturbed adjacent strata, the initial methane content of the seams, and the proposed dimensions and weekly advance rate of the face (Dunmore, 1979).

The MRDE prediction method uses Airey's theory to determine the "degree of gas emission" functions for gas which enters the workings from each of three sources: coal seams in adjacent strata, the coal seam being worked, and broken coal on the transportation system. The value of the emission function for coal seams in adjacent strata is dependent upon the depth of the workings, the age of the district, and the distance between the methane source and the workings. The value of the emission function for the coal seam being worked is dependent upon the rate of advance of the face. The degree of emission from broken coal on the transportation system is applied to the gas content still remaining in the coal at the face and is a function of the time required for broken coal to be conveyed out of the district (Dunmore, 1981).

MRDE predictions of methane emission have been compared with measured emissions in longwall workings, and have been found to be within about 20 percent of the measured levels. Further investigation is needed to quantify the effect of the nature of the strata on methane migration paths as this influences the relative emissions from the roof and floor (Dunmore, 1979).

In addition to quantifying the emission rate from each methane source, the MRDE estimation method provides for calculation of the volume of air required to dilute the emitted methane to acceptable levels of concentration. In order to calculate the required air quantities, it is necessary to assume, or to determine through investigation, the manner in which the gas from specific sources migrates into the airways or methane drainage boreholes. To ensure that the calculated airflow rates will be adequate, the method is based on the results of a statistical analysis of peak rates of emission as indicated by recordings of airway methane concentrations.

5.2 HOMOTROPAL VENTILATION

In general, mine workings are ventilated ascensionally rather than descensionally to take advantage of the natural ventilation pressure that is created by the reduction in density of the air as it is heated during its passage through the mine workings. This effect tends to cause the air to flow from the lower to the higher levels, which assists the pressure produced by the fan if the ventilation of the workings is ascensional. Under these conditions, if the fan is stopped, the ventilation will continue to flow in the normal direction but at a reduced rate. If the ventilation is descensional, however, the natural ventilation pressure developed in the workings opposes the pressure from the fan and the natural ventilation pressure in the shafts; stopping of the fan then causes a greater reduction in the airflow through the mine, and may cause a reversal of the airflow in the workings.

The continual increase in the depth of mine workings, together with increased mechanization involving higher installed power, have resulted in environmental problems due to dust, heat, and humidity. In consequence, further consideration has been given to alternatives to ascensional ventilation. The term "homotropal" was coined by Bromilow (1957) for the system of ventilation in which air and coal flow in the same direction, i.e. the coal is conveyed in the return. As Bromilow was careful to point out, the term is not synonymous with "descensional ventilation" though, where the gradient along the face line is at all severe, the need to convey down the gradient would require a homotropal system to be also descensional. The term descensional ventilation is therefore generally used to describe a ventilation airflow which is both descensional and homotropal. Ventilation is antitropal when the ventilation air and the coal flow in opposite directions. The advantages and disadvantages of these ventilation systems are considered in relation to airborne dust, methane, climate, and operational factors.

5.2.1 Airborne Dust

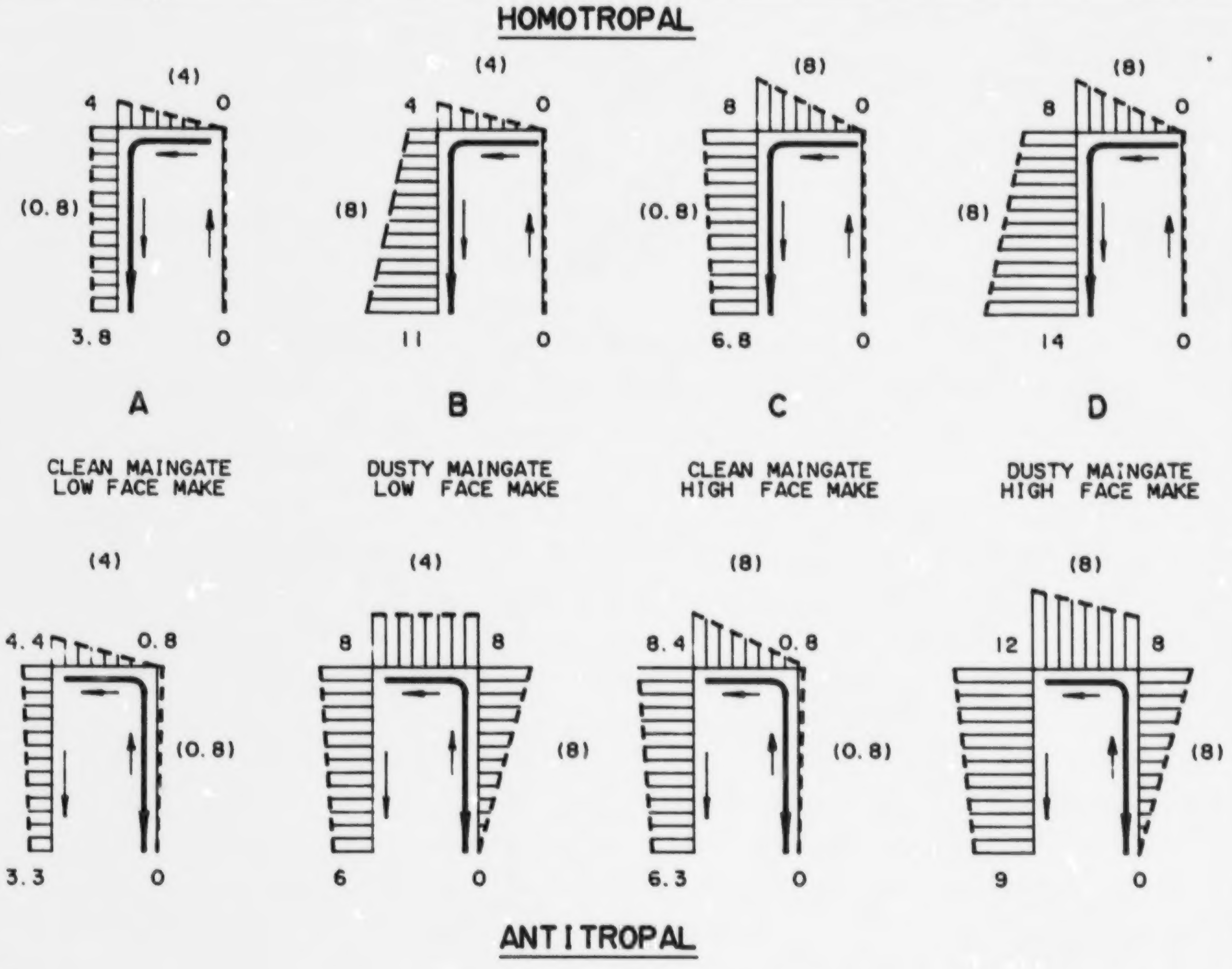
The total dust concentration at the return end of a longwall face may be taken as the sum of the contributions of each source of dust in the normal operation of the face. It is generally considered that, in antitropal conditions, intake air contamination arising from air/conveyor relative velocity, face to conveyor transfer points, face end activities, and increased stone dusting in conveyor roadways, will be greater than the contamination experienced under homotropal conditions. In these circumstances Ford (1974) considered that homotropal ventilation would reduce the airborne respirable dust carried from intake to return. The following page provides a diagrammatic representation of the effect of homotropal and antitropal ventilation on dust concentrations. Homotropal ventilation results in lower dust concentrations than antitropal along the intake, the coal face, and the inbye end of the return airway. (National Coal Board, 1977) The disadvantages of antitropal ventilation are reduced if measures are taken to minimize dust pollution in the conveyor roadway, but under all conditions homotropal ventilation reduces the dust concentration on the coal face.

5.2.2 Methane

The emission of methane from the coal on the conveyor in the intake airway may only be significant where a long conveyor is transporting a large tonnage of gassy coal and the methane content at the inbye end of the antitropal intake airway could approach statutory limits (National Coal Board, 1977). The main consideration is that, in inclined seams, conveying in the lower face roadway will require the use of descensional ventilation. Both Bromilow (1957) and Carver (1960) agreed that descensional ventilation would not significantly affect the total gas emission in the district, but the buoyancy of the methane in the gob, in opposition to the applied ventilating pressures, would result in the gas fringe approaching closer to the face line and more methane appearing at the return end of the face. Johnston and Tabor (1972) considered that, if homotropal ventilation was to be established, there should be a generous margin available in the ventilation system.

Other dangers inherent in descensional ventilation are:

- the risk of roof layering if air velocity is inadequate
- ventilation reversal in the event of a fire on the face or a loss of applied ventilation pressure


HOMOTROPAL
(4)
4
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(0.8)
3.8
0
A
CLEAN MAINGATE LOW FACE MAKE
(4)
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0
(8)
11
0
B
DUSTY MAINGATE LOW FACE MAKE
(8)
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0
(0.8)
6.8
0
C
CLEAN MAINGATE HIGH FACE MAKE
(8)
8
0
(8)
14
0
D
DUSTY MAINGATE HIGH FACE MAKE
(4)
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(0.8)
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0
(8)
12
8
(8)
9
0
ANTITROPAL

DIAGRAMMATIC REPRESENTATION OF THE EFFECT OF HOMOTROPAL AND ANTITROPAL VENTILATION ON DUST CONCENTRATIONS

LEGEND

COAL TRANSPORT

VENTILATION

DUST CONCENTRATION

(8) DUST MAKE

IN THE FOUR EXAMPLES ILLUSTRATED THE FOLLOWING CONDITIONS APPLY:

THE AIR INTAKE IS CLEAN
A CLEAN MAINGATE GENERATES 0.8 UNITS OF DUST
A DUSTY MAINGATE GENERATES 8 UNITS OF DUST
A LOW FACE MAKE OF DUST IS 4 UNITS
A HIGH FACE MAKE OF DUST IS 8 UNITS

IN EACH CASE THE GENERATION OF DUST IS UNIFORM ALONG THE MAINGATE OR FACE.

IN PASSING THROUGH THE FACE ONE HALF OF THE INTAKE DUST IS LOST.

IN PASSING THROUGH THE RETURN GATEROAD ONE QUARTER OF THE INBYE DUST IS LOST.

NOTE: THE VALUES USED ARE PURELY ARBITRARY, AND FOR ILLUSTRATION PURPOSES ONLY.

Taken from N.C.B. Mining Environment Bulletin No. 1
N.C.B. Mining Department Nov. 1977

- the highest methane concentrations occurring in the conveyor roadway where most of the machinery is situated

5.2.3 Climate

In considering the influence of homotropal ventilation on mine climate, Varma (1980) concluded that the climate in mine workings deteriorated as face output increased due to an increase in freshly exposed strata, increased machine power producing heat, and increased heat emission from roadway strata and conveyed coal. Experimental results indicated that, with a homotropal system, there is a general reduction in air temperatures particularly on the intake side.

The National Coal Board (1977) noted that at corresponding points in the ventilation circuit, the Effective Temperature as defined in the American Effective Temperature Scale will almost invariably be one degree or more lower with homotropal than with antitropal ventilation. At an Effective Temperature of 28°C a difference of 1°C represents a very significant change in the comfort level. It is generally concluded that climatic conditions at the inbye ends of a longwall district are improved by homotropal ventilation, although the conditions at the outbye end of the return airway will not be significantly changed.

5.2.4 Operational Considerations

The main concern over the introduction of homotropal ventilation is the problem of underground conveyor fires. In the return, a conveyor fire would have to be attacked via the face and arrangements made for fire-fighting water pipes through the face. Johnston and Tabor (1972) did not appear to rate this a serious problem in describing the homotropal layout at Monktonhall Colliery. An advantage of this system is the fact that, in the event of a conveyor fire, most of the men would be upwind and therefore at less risk than from a conveyor fire in the intake.

5.2.5 Conclusions

It is considered that homotropal ventilation may:

- offer some reduction in dust concentration on the coal face
- have minimal effect in reduction of methane concentrations
- increase the hazards associated with higher gas concentrations in roadways with electrical equipment
- improve climatic conditions
- pose fire-fighting problems for conveyor fires in the return counter-balanced by increased safety for men at coal face (for more detail, refer to Section 7.3 concerning fires on main conveyor roads)

5.3 ASPECTS OF GENERAL MINE PLANNING AFFECTING VENTILATION

This section briefly considers some aspects of general mine planning and their relationship to ventilation requirements. Later portions of Section 5 deal with planning for development drivages, room and pillar, longwall (advance and retreat), hydraulic mining, etc., with comments on the alternative layouts for specific mining methods.

5.3.1 Straight Through Ventilation

Many mines experience ventilation problems because main intakes and returns parallel each other, and are connected by frequent cross-cuts which may have been necessary for operation of equipment and ventilation of headings during drivage of main developments. Once the layout is established, the cross-cuts become a liability, particularly if high ventilating pressures and/or weak ground around the perimeter of stoppings encourage leakage.

In some situations a system of straight-through ventilation can be established, whereby intake and return slopes or shafts are located on opposite sides of the mineable reserve. A significant reduction in leakage, and a corresponding reduction in ventilation efficiency, can be achieved with this system but such a layout is not practicable where access is limited, as in rugged topography or submarine mining. It has application in the mining of level seams where the depth and ease of access is not particularly variable over the mineable area.

5.3.2 Elimination of Stoppings

In many installations in North America mains have been developed in two sets, with two to five headings in each set. One set of headings is used for intake air, the other for return ventilation. A solid barrier of coal is left between the first and second set of headings. The barrier pillar serves to greatly reduce leakages and may improve strata conditions in the panel.

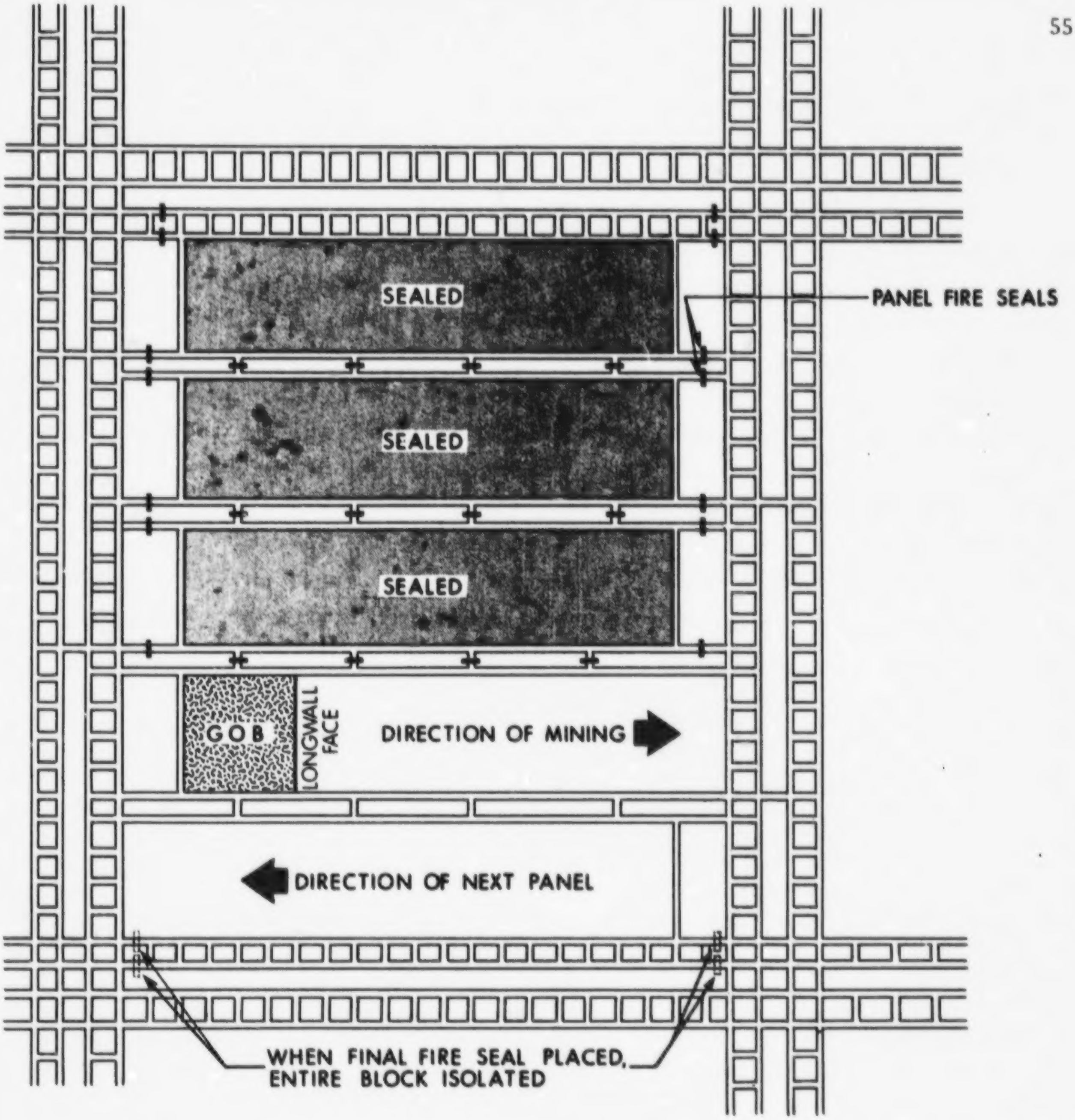
5.3.3 Airway Cross-Section

The adequacy of the total cross-section of intake or return airways is critical in any mine. For any given volume requirement, there is a choice between a larger number of small airways, and fewer airways of greater cross-section. Factors to be considered in the final decision include:

a) Thickness of the coal seam
b) Security of individual roadways at various widths
c) Anticipated cost of roadway maintenance
d) Security of the entire panel when developed to various configurations
e) Other services required, such as transportation of coal, men and materials; emergency exit routes, etc.; and compatibility with ventilating circuits

5.3.4 Blocking Off

Benefits are often gained by separately developing individual panels, for either longwall or room and pillar operations. Elimination of ventilation leakage, reduction of heating risks and ease of sealing off in case of fire are some of the more obvious advantages. Figure 5.2 shows a layout for retreating longwall which is commonly used in the United States. This layout has the following advantages:


SEALED
PANEL FIRE SEALS
SEALED
SEALED
GOB
LONGWALL FACE
DIRECTION OF MINING
DIRECTION OF NEXT PANEL
WHEN FINAL FIRE SEAL PLACED, ENTIRE BLOCK ISOLATED

LAYOUT OFFERS OPTION TO EMPLOY BLEEDER SYSTEM OF RETURNS.

BLOCKING OFF LAYOUT IN COAL LIABLE TO SPONTANEOUS COMBUSTION

FIGURE 5.2

NorWest
Resource Consultants Ltd

a) In a gassy seam, the use of bleeders can help to keep the gob edge and face clear of methane

b) If the seam is liable to spontaneous combustion, the bleeder returns can easily be eliminated by suitably placed seals

c) Each panel, or the entire block, can be sealed quickly in the event of heating

d) Good ventilation of the development drivages is possible through the use of a two-entry system and the possibility of driving entries from either side of the block.

Disadvantages of this layout include:

a) Reserve recovery is rather low, even if some pillar coal can be extracted

b) The ratio of development to longwall coal is high

c) If pillar recovery is ineffective or is not attempted, subsidence around the perimeter of the block is uneven, which may be a serious disadvantage in multi-seam mining or beneath sensitive surface areas.

5.4 AUXILIARY VENTILATION OF DEVELOPMENT DRIVAGES

Development drivages requiring auxiliary ventilation fall into three main categories:

a) Main drivages, to form the underground access for ventilation, transportation and other services. These main drivages may be in the coal seam(s) or in rock. Extended drivages are usually developed as a group of two or more, with auxiliary ventilation carried forward and limited to the inbye end of the advancing system.

b) Submains that block off the resource into logical mining units and are usually driven as groups of two or more headings. Auxiliary ventilation is carried forward with the advancing development system and limited to the inbye end because long single headings are not involved.

c) Panels, which may be formed by single headings or by a multi-heading system of development.

Very long, single headings are a difficult ventilation problem when gassy or dusty conditions are encountered. However, some coal operators prefer the long single heading method of longwall retreat panel development. A panel length of 2000 metres for example, presents a ventilation problem where sufficient volume has to be maintained at the face and environmental conditions have to be kept within prescribed limits. One way of reducing the difficulties associated with a long single heading drivage is to block off the resource area with submains and develop the heading from both sides, thus reducing the ventilation and transportation lengths, but this is not always feasible.

Factors that influence the ventilation of a long single heading, and subsequently the rate of development, include:

a) total projected length of heading
b) cross- sectional area of heading
c) depth of heading from surface
d) method of ventilation:
    exhausting auxiliary
    forcing auxiliary
    overlap auxiliary

e) type of equipment and mining method
f) presence of water
g) methane emission rate and dust.

Development drivage faces are ventilated by an auxiliary fan system or by line brattice directing to the blind ends a proportion of the general ventilating intake air split coming into that particular district. Occasionally a combination of the two is utilized.

5.4.1 Use of Line Brattice For Auxiliary Ventilation

Modern brattice is manufactured from flame-proof plastic or nylon and supplied in rolls of appropriate length and width (height). When suspended along a blind heading from the roof, the brattice divides the excavation into compartments, one forming a passage for intake air and the other for return. The brattice line starts at a place where the ventilation coming into the heading system can be intercepted causing a proportion of that air to be diverted into the blind ended excavation.

Brattice line ventilation of headings is commonly used in the United States where much skill in using the system has been developed. Many operating crews prefer the system over auxiliary fan and tubing methods, to avoid the high noise levels of a fan. With good supervision and a well-designed system, brattice line auxiliary ventilation can aid in complying with dust limits of 2 mg/m^3 as determined from personnel dust samplers worn by face crews and others.

In a blind heading 5 metres wide, the line brattice might divide the excavation into two passages one side 4 metres wide (to allow free movement of mining equipment) and the other side 1 metre wide.

Up to a few years ago, normal practice in the U.S.A. and to a lesser extent in Western Canada, was to force intake air into the heading up the narrow side formed by the line brattice. The intake air would scour the face of the heading and return over the mining equipment and the crew. When the statutory limits for respirable dust were tightened in the U.S.A., the system was reversed, so that the intake air courses up the wide side of the heading and passes over the crew and face equipment, taking the dust from mining operations directly into the return on the narrow side of the line brattice. This change enabled dust counts to be maintained within statutory limits, but only if diligence was exercised by the operating crews and supervisors. To prevent methane layering and dust roll-back over the mining equipment, the line brattice had to be kept within approximately 3 metres of the face, and air volumes had to be maintained to an appropriate and consistent level.

Advantages of the system include:

a) Silent in operation.
b) Does not require electrical or mechanical equipment.
c) If the main fan is stopped the system can be left in place, during and after the stoppage. (The electrical power to an auxiliary fan is turned off if the main fan stops.)
d) Ventilation continues as long as the main fan is in operation, (including week-ends).
e) Ventilation can be adjusted and rearranged relatively quickly following new cross-cut breakthroughs.

Disadvantages of the system include:

a) In a very gassy situation, the low turbulence in the air flow may permit gas to concentrate at the face.
b) It is only suitable for headings up to about 30 metres long.
c) It is difficult to prevent leakages resulting from sub-standard installation.
d) The brattice is vulnerable to damage and ventilation interruption if equipment and personnel bump into the brattice line. Leakages of up to 75 percent of incoming air are not uncommon in poorly maintained brattices .
e) More cross-sectional heading area is sacrificed with a line brattice system than with ventilation ducting.
f) It is difficult to erect and maintain in mining heights over 2.5 m.

5.4.2 Auxiliary Ventilation By Fans and Ducting

Note The types of fans and ducting available are described in Section 4.3.

There is no single method of auxiliary fan ventilation which is superior in every respect. A decision as to which type of system is most appropriate has to be made on a site specific-basis. If dust is the major concern, the method of inbye ventilation might be different than if the major problem was methane emission rates or working temperatures at the face. More use is now being made of combination methods of auxiliary ventilation. These provide systems which overcame some of the problems that cannot be handled by a single method. The systems which can be applied are described briefly below.

a) Exhausting

This method has the advantage that airborne dust is not only drawn from the working face but is kept within the confines of the ducting, thus keeping the total heading relatively dust free. On the other hand, where a long single heading is driven within a coal seam, methane from coal ribs is carried to the working face where electrical equipment is being operated. Other disadvantages include higher temperatures at the face, and less scouring of the face area by the air stream. Methane concentrations in the return air within the ducting can reach levels so high that the fan may have to be shut down. Should methane layering occur, the layers are not easily broken up by using the sucking action from the ducts.

b) Forcing

In a forcing system, relatively fresh, cool air is discharged from the outlet of a duct. This air scours the face and turbulence prevents methane accumulations in the corners. Methane from coal ribs is carried away in the air returning from the face. Any methane layers along the heading can be broken up by tees from the forcing duct blowing air out at roof level. The major drawback of this system is airborne dust. The air discharged from the end of a duct close the face picks up the dust created by mining equipment, at the face and rolls it back over the equipment and face crew. In theory, the problem can be overcome by good dust suppression at the face, but in practice, this is not easy to achieve because of the difficulty of "wetting" coal dust.

c) Overlap Systems

Overlap systems include exhausting with a forcing overlap or forcing with an exhausting overlap. Authors who describe the system invariably warn against incautious use of overlap systems due to the dangers of re circulation. However, properly designed and operated overlap systems have advantages that simple forcing and exhausting do not offer. For example, a main auxiliary exhausting system, which is supplemented by a short inbye forcing system, draws airborne dust directly into the main exhaust duct while at the same time providing necessary turbulence at the face. It is essential with overlap systems that all the fans in the system are interlocked so that, if the outbye fan is stopped for any reason, the overlap fan will not continue to run. If this happened, methane could build up in the face area and be re-circulated by the overlap fan.

d) Controlled Recirculation

Controlled recirculation, if carefully applied, can be effectively employed to help overcome certain difficulties. The build-up of methane, which can occur with a recirculating system, will reach a condition of equilibrium at a pre-estimated, maximum, safe concentration. Careful control and monitoring is necessary to ensure that the condition of equilibrium is not reached at an unsafe level. (National Committee Report. 1977).

Controlled recirculation is commonly employed in the following situations.

i) Where an exhausting auxiliary fan and dust scrubber are used to deal with dust at the face of an advance heading at the intake end of an advancing longwall. As the auxiliary system exhausts into the intake, a small portion of exhausted air is drawn into the heading face.

ii) Where a forcing auxiliary system is used to ventilate the face of an advance heading at the return end of an advancing longwall.

The air drawn into the fan and forced to the face of the heading will contain a small portion of the air which has just left the heading and joined the return air leaving the face by the return roadway.

Note: Prevention of recirculation in either of the above cases would require the fans to be located, impractically, on the face line.

iii) Where a long heading is ventilated by a forcing auxiliary system in conjunction with an exhausting overlap system at the face to deal with dust control. Conventionally, the exhausting overlap would only circulate 50% of the air supplied by the main forcing fan in order to prevent recirculation. In cases where a very long, single heading has to be ventilated, the volume of air needed to provide the 50% excess, the high water gauge required, and the air volume needed to compensate for duct leakage, can approach practical limits. In these cases controlled recirculation in the overlap section can be used to overcome the difficulties. However, it is important to ensure that roadway air velocity in the overlap section will be sufficient to prevent layering. (Colloquim on Mine Ventilation, Paper No. 3 1970).

e) Two other systems used in appropriate circumstance:

i. Standby fan set in parallel: In this arrangement, two fans of identical characteristics provide air through a Y-piece to a common duct. Butterfly valves direct air flow from either one or both of the fans into the common duct. This arrangement provides a standby source of ventilation if one fan fails. If both fans are operated together there is a substantial increase in volume, making the arrangement useful in headings that have high methane emissions and where fumes need to be expelled quickly after shot firing. (Barker and Highton, 1981)

ii. Reversible System: This system is particularly useful for ventilating long rock drivages where multi-shot blasting of a large number of charges is used to advance the face. If the heading is ventilated by a forcing auxiliary system, considerable operating time can be lost waiting for the blasting fumes to slowly travel the whole length of the tunnel. By changing the ventilation, temporarily, into an exhausting system, the fumes are extracted much more quickly through the ducting. One way of accomplishing this change is to use rigid ducting which bifurcates at the outbye end, one limb to the upstream of the main air flow and the other down stream. In the upstream limb is the normally operating forcing fan and in the downstream limb, the exhausting fan. Each limb can be closed off and the two fans are interlocked so that they cannot both be operated at the same time.

5.4.3 Auxiliary Ventilation By Air-Movers

The air-mover relies on the venturi principle. Using this system, a flow of compressed air issues through a jet, placed in the throat of a duct to induce a much larger flow of free air through the duct. It is common to achieve a free flow of five to eight times the quantity released by the jet. The air-mover may be used for auxiliary ventilation of a heading. In a typical application an air-mover made from 18 mm tubing, shaped into a ring and having 9 mm holes drilled at intervals around the ring, is placed within the duct (Barker 1981). The jets issuing from the ring induce air flow of 2 m^3/s through the duct, using only 0.25 m^3/s of compressed air. In this type of installation, an air-mover could be utilized either:

a) As the sole source of auxiliary ventilation over a short distance

b) As a back up for an auxiliary fan, or as a replacement when the face is idle (eg. at a weekend)

c) To continue ventilation of the face of a heading which is not being advanced, where both methane and dust emissions are low.

Air-movers are also useful for dispersing small quantities of rich methane/air mixtures in locations which cannot be cleared by fan-induced ventilation, such as the corners of longwall face ends and the face area of headings (Colloquium on Mine Ventilation, Paper No. 31970).

5.4.4 Miscellaneous Aspects of Auxiliary Ventilation

The type of transportation system used in a heading has an effect on ventilation efficiency. A low-profile belt conveyor system, with a minimum of transfer points, will generally cause less resistance and turbulence than LHD trucks or trains of cars. Heavy duty roadheaders, full-face tunnel boring machines, and continuous miners will give off more heat and cause greater obstruction to air circulation than will a slusher bucket loading out a round of muck. The humidity in a heading will be increased appreciably where heavy-duty machines use numerous banks of dust suppression sprays supplied by water piped through water cooled motors.

Airborne dust in long single drivages is a serious problem. Respirable dust from rock cutting operations presents a direct physiological danger to exposed personnel. Fine dust from coal cutting operations represents both a physiological and an explosion hazard.

Control of airborne dust is a complex problem and cannot be addressed comprehensively in this report. Dust extractors on continuous miners are becoming common on the coal getting equipment offered by the various manufacturers, but care must be exercised with this type of apparatus. For example a continuous miner with a dust extractor drawing ventilating air around the cutting heads could dilute high concentrations of methane to an explosive mixture where a spark from the cutting picks could result in ignition. Dust extractors should therefore be capable of maintaining sufficient through-put to keep methane levels below the explosive limits. Additionally, the extractor intakes should be designed so that they cannot become blocked with coal or other debris during operation.

5.5 VENTILATION CONSIDERATIONS SPECIFIC TO ROOM AND PILLAR OPERATIONS

This section considers:

a) Development of multiple headings as mains and submains.
b) Development of room and pillar sections.
c) Extraction of pillars.

Ventilation of a room and pillar mining system is generally more complex than ventilation of a longwall panel. The complexity arises from the cyclic operation of a continuous miner section, the miner, shuttle cars and rock bolter constantly moving from face to face during the shift. Each Miner section is usually required to be ventilated by a separate air split; each room (or heading in a multi-heading development) has to be provided with ventilation at the face. In addition, certain rooms or headings must accommodate shuttle cars, feeders, conveyors and supplies transportation. Room and pillar operations can be divided into two phases: development, and depillaring.

5.5.1 Development Phase

Because room and pillar has a less favourable development to extraction ratio than longwall mining, more mining sections are needed to achieve the same production. For example, if one longwall unit mines 4500 tonnes per day, and is supported by three continuous miners on development, then total daily tonnage for the whole mine would be 6300 tonnes (Assuming an average production of 300 tonnes per miner per shift on development and 2 shifts per day). Such a mine would contain four separately ventilated mining districts. If the same daily production was mined from a room and pillar mine with an average production from both development and pillar sections of 400 tonnes per shift, then the number of ventilated mining districts would be eight, based on two shifts production daily.

From the above example it can be demonstrated that, in order to ventilate eight working sections with separate splits of air and maintain a low ventilating pressure, at least six main airways are required, particularly if the main coal transport road or belt road has to be ventilated at a reduced air velocity. To facilitate rapid place changing when these multi-heading systems are developed, requires the formation of cross-cuts between headings at frequent intervals. These cross-cuts are a source of leakage for the life of the mine.

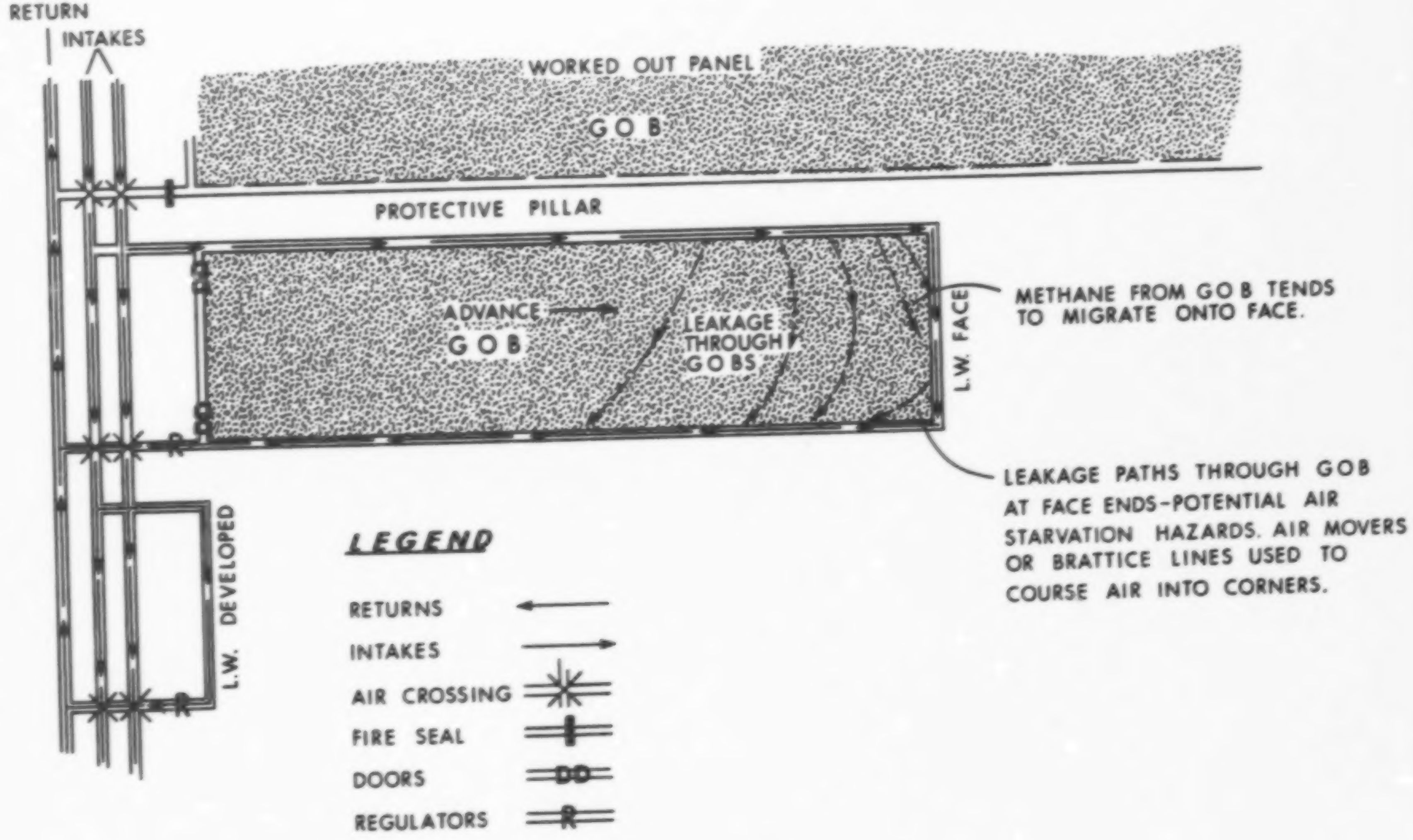
5.5.2 Depillaring Phase

The bleeder system of ventilating a pillar section allows the intake air coming into the face area to be directed through the gob and into returns at the rear of the panel. When successfully employed, this method substantially reduces airborne dust concentrations in the working sections and reduces loss of intake air from leakage. If the panel headings can be kept partially open through the gob by the installation of a row of wooden chocks, then stagnant zones, that allow methane buildup, will not form along the pillar line and in blind corners.

Where a bleeder system of returns is not employed (for example where there is risk of spontaneous combustion or where it is not practicable to develop submains around a block) then an alternative method of ventilation must be used. Usually, fresh air enters at one end of a pillar line and, after traversing the workings, flows through the panel returns at the opposite side of the panel. On a well designed and operated layout, dust and methane concentrations do not usually occur. However, float dust in the panel returns might require large quantities of stone dust to maintain a prescribed level of noncombustibles at the roof, sides and floor. If methane emissions are high, greater air quantities will be necessary along the pillar line. This is a disadvantage compared to the bleeder system of ventilation where any extra air would be directed up the bleeders rather than through the pillar line itself.

5.6 VENTILATION CONSIDERATIONS SPECIFIC TO LONGWALL ADVANCING SYSTEMS

A typical advancing longwall operation with the panel conveyor placed in intake air is shown in Figure 5.3. The following effects occur with such a layout:


RETURN
INTAKES
WORKED OUT PANEL
GOB
PROTECTIVE PILLAR
ADVANCE
GOB
LEAKAGE THROUGH GOBS
L.W. FACE
L.W. DEVELOPED
R
LEGEND
RETURNS
INTAKES
AIR CROSSING
FIRE SEAL
DOORS
REGULATORS

METHANE FROM GOB TENDS TO MIGRATE ONTO FACE.

LEAKAGE PATHS THROUGH GOB AT FACE ENDS-POTENTIAL AIR STARVATION HAZARDS. AIR MOVERS OR BRATTICE LINES USED TO COURSE AIR INTO CORNERS.

TYPICAL VENTILATION FOR AN ADVANCING LONGWALL PANEL

FIGURE 5.3

a) Methane desorption from coal ribs, adjacent gob, and from the coal stream on the conveyor system itself will cause some intake air contamination before it reaches the face.

b) If the system of ventilation is antitropal (See Section 5.2), appreciable quantities of airborne dust will be picked up from conveyor rollers, exposed roadway surfaces, and from between separation doors along conveyor roads. Part of this dust, carried in the intake air, will be added to by coal dust generated at the stage loader and face conveyor transfer points before it is carried onto the coal face. However, if the ventilation is homotropal, the dust from the above sources is directed away from the longwall face because the belt conveyor system would then be in the return airway.

c) On a fully advancing longwall panel not using a cundy system (See Section 5.8), the total district ventilation volume reaching the face will traverse the wall itself. This can lead to high air velocities, particularly where gassy conditions require high volumes. If methane emission rates are particularly high, production can be inhibited due to limitations placed on face air velocities. In a recent feasibility study a Canadian, gassy, underground mine was to use longwall as the principal method of extraction. Even though the proposed method included retreat mining with a bleeder system of returns and a comprehensive methane drainage arrangement, production levels were still restricted to well below equipment capability due to a 4.5 m/s air velocity restriction on the faces.

The panel intakes and returns leading to an advancing longwall face run parallel and adjacent to the extracted area. The pressure differential between them commonly causes air leakages across the gob to reach serious proportions. The problem increases as the panel penetrates deeper into the block. As much as 60 percent of the total air delivered to an advancing longwall panel can be lost to leakage. The problem is often exacerbated by increasing the district pressure and ventilating volume to compensate for the leakage and provide adequate face ventilation. Methods of reducing this problem are outlined in Section 5.8.

The pattern of air flow through an advancing longwall face can result in blind ends being inadequately ventilated, allowing methane buildup. Ventilating these face corners is a critical factor in maintaining a safe environment and, if face end designs cannot avoid the problem, a method of mechanically providing corner ventilation might have to be installed.

Roadway cross-section reduction as a result of convergence occurs to some extent on all longwall panels, but more particularly on advancing systems. Roadway distortion and floor heave increase the resistance to panel ventilation, resulting in lower quantities reaching the face unless increased pressure is available. With rapidly advancing longwalls, the need to speed-up drivage of the headings has led to a congestion at the roadhead. Equipment is utilized which, at times, severely restricts air flow, creating zones of high air velocity at each end of the face run. With high air velocity, coal dust is raised, and carried along with the face ventilation.

5.7 VENTILATION CONSIDERATIONS SPECIFIC TO RETREATING LONGWALL SYSTEMS

Retreating longwall faces have many operational advantages over advancing systems. The most important are:

a) Delineation of geological features during development.
b) Completion of entry drivage during development reduces some of the operational delays associated with the face ends of an advancing longwall.
c) More scope to control outbreaks of spontaneous combustion.

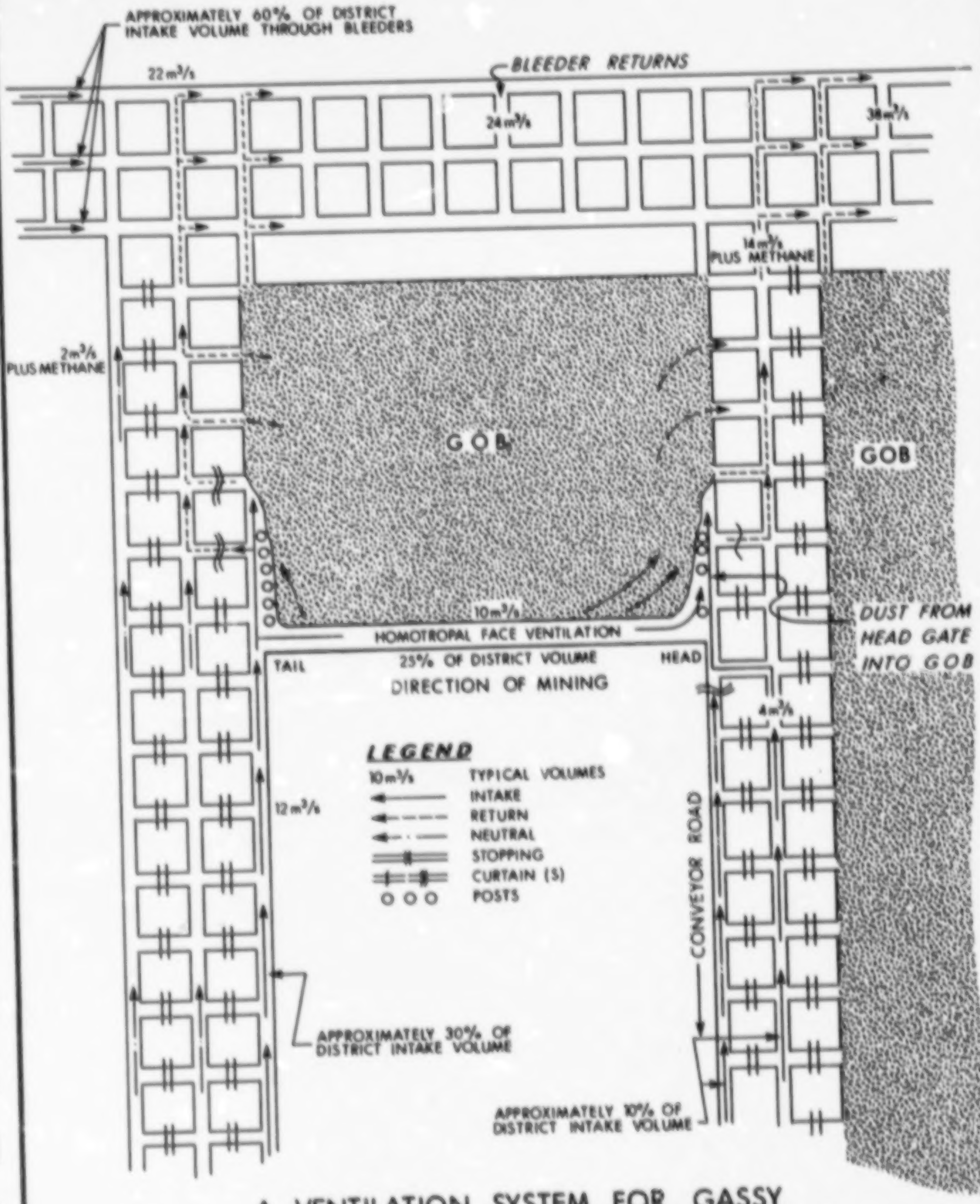
During the extraction of gassy seams using a simple retreating longwall system there is a tendency for methane/air mixtures, in the unconsolidated gob close to the face line, to migrate into the face area at the return end of the wall. The ventilating pressure distribution on a retreating face encourages this effect. If the face is retreating to the rise in a dipping seam, the natural buoyancy of the methane/air mixture will exacerbate the situation. Migration of gas from the gob at the return end of the face is particularly dangerous since the ventilating air at this point already contains the methane emitted from the freshly exposed coal over the full length of the longwall. Some of the electrically powered face equipment is almost invariably located at the return end, creating potentially hazardous conditions. Generally speaking, the situation is worse on this simple type of retreating face than on an advancing longwall. In the latter case part of the ventilating current tends to sweep the open gob area behind the supports, keeping it clear of dangerous concentrations of methane as long as the roof caves regularly and the open area is not too extensive.

The difficulty of maintaining a safe environment on retreating faces has led to the adoption of a bleeder system of ventilation in retreating longwall. A typical layout is shown in Figure 5.4. In this system the entries at both ends of the face, outbye the face line, carry intake air. An intake air split is taken from the main intake entry up the longwall face. The entries on the gob side of the face act as returns. The air flow is such that the open edges of the gob, including the corners at each end of the face, are swept by air which then passes through the entries on the gob side of the face to the bleeder returns. A flow-through system such as this is very effective in reducing the hazards caused by gas flushed from the gob and is therefore very effective where methane emission rates from the seam and adjacent strata are high. The use of bleeder systems is mandatory in the United States; exemptions are difficult to obtain. In Canada, the only retreating longwalls which have operated recently have been at the Prince Mine in Cape Breton where gas emissions were very low and bleeders are not used.

Two options are available in using bleeder returns:

a) Either the bleeder entries on one or both ends of the wall, alongside the gob and inbye the face, are maintained so they can be traveled and inspected, or

b) The bleeder entries at the gob side of the face are not maintained and kept open for inspection, though supports may be set to keep the abandoned entries open sufficiently to permit the flow of return air.


APPROXIMATELY 60% OF DISTRICT INTAKE VOLUME THROUGH BLEEDERS
22 m³/s
BLEEDER RETURNS
24 m³/s
38 m³/s
14 m³/s PLUS METHANE
2 m³/s PLUS METHANE
GOB
GOB
DUST FROM HEAD GATE INTO GOB
10 m³/s
HOMOTROPAL FACE VENTILATION
TAIL
25% OF DISTRICT VOLUME
HEAD
DIRECTION OF MINING
4 m³/s
LEGEND
10 m³/s TYPICAL VOLUMES
INTAKE
RETURN
NEUTRAL
STOPPING
CURTAIN (S)
○ ○ ○ POSTS
12 m³/s
CONVEYOR ROAD
APPROXIMATELY 30% OF DISTRICT INTAKE VOLUME
APPROXIMATELY 10% OF DISTRICT INTAKE VOLUME

A VENTILATION SYSTEM FOR GASSY RETREAT LONGWALL WITH BLEEDER RETURNS

FIGURE 5.4

The first option is attractive in gassy coal because the maintained entries permit methane drainage using cross-measure boreholes in an area where large quantities of methane can be tapped.

The use of either type of bleeder system does introduce some technical problems.

a) The ratio of development to longwall extraction tonnage is appreciably increased.
b) The risks of spontaneous combustion are significantly increased, because air flow through the gob, and through return entries subject to intensive crushing, is required.

In a seam prone to spontaneous heating, the use of a bleeder layout may be impossible, particularly a system in which the entries are not kept open for inspection. Under these circumstances, the use of a cundy to provide a back-return system should be considered. This layout is described in detail in Secton 5.8.3 and provides a means of controlling any flushing of gas onto the return end of a retreating longwall face without the increased risk of a heating induced by a bleeder return system. Access for methane drainage is also provided by the back-return.

Several case histories of retreating longwalls were provided by Highton (1979). These describe the use of maintained entries, nonmaintained entries and back-return systems. They illustrate in detail the points covered above.

5.8 CUNDY VENTILATION

5.8.1 Introduction

A cundy, or snicket, was originally defined as a passage from a ventilated roadway, through the adjacent ribside pillar or packwall, giving access to the gob area of a longwall panel. In recent years, the term has referred to a passage driven from a ventilated roadway, through an adjacent coal pillar or packwall, to the roadway of a previous panel.

The initial purpose of the cundy was to assist the flow of gas from the gob, for dilution in the ventilating current of the roadway. By using a cundy, dangerous accumulations of gas in the gob were reduced and the tendency for methane to be flushed onto the face by caves or changes in barometric pressure was reduced. In investigations into flushings of gas from the gob at Markham Colliery (Davies, et al., 1969), it was noted that cundy holes were not capable of conveying significant amounts of gas through the return gate pack. In a colloquium on mine ventilation (Institution of Mining Engineers 1971) in considering the face end configurations of longwall faces, it was concluded that:

a) ribside cundies into the face often do not adequately improve the ventilation of the face ends
b) methane layering may occur in the cundy due to low velocity of airflow
c) this method may have disadvantages in seams liable to spontaneous combustion.

In advancing longwall workings, the quantity of air reaching the face may be severely reduced by leakage between intake and return across the gob outbye of the face. Lomax (1964) reported experiences when advancing a 137 m longwall face for 300 m then for an additional 200 m at a face length of 82 m. Airflow at the intake outbye end was 15.4 m^3/s, whilst 3.6 m^3/s entered the face, so that gob leakage flow accounted for 77 percent of total airflow in 500 m of face advance. The provision of a cundy into the gob through the return roadway pack did little to reduce a continuous increase in methane concentrations at the face return end. The face ultimately suffered a major outburst of methane from the floor in which the emitted gas completely blocked face ventilation and all of the 14.2 m^3/s of air which entered the intake circulated through the gob.

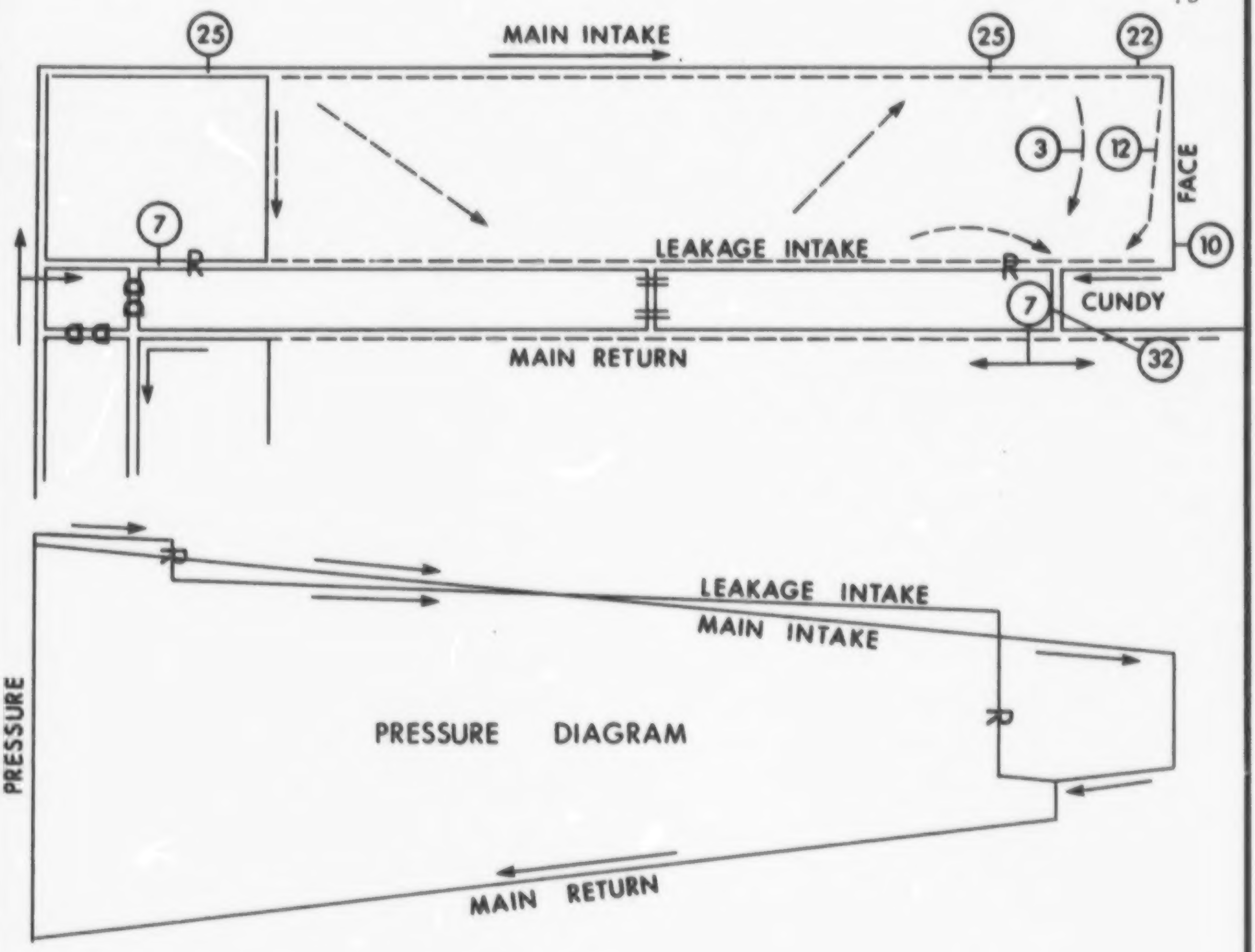
In addition to the problems of methane dilution, airflow through the gob may induce or sustain spontaneous combustion. Gob leakage may be reduced by applying sealants to the packwalls and/or by increasing the density of packing in an attempt to increase the ventilation resistance of the leakage paths. However, such methods have practical and economic limitations and it is more desirable to minimize leakage by reducing the ventilation pressure difference between the face roadways. In effect, both roadways become intakes and a new return is established through an adjacent disused "receiving" face by a connecting cundy.

5.8.2 Uses in Longwall Advancing Faces

As described above, cundies have been used to connect an existing ventilated roadway through a coal pillar or packwall to a previously driven roadway. In the case of a longwall advancing panel and pillar layout, a cundy may be used to connect the return roadway to the disused roadway of the adjacent panel by holing through the intervening coal pillar. By using the adjacent panel as the return airway, both roadways of the active panel may become intakes with the former return being tightly regulated as a leakage intake. After passing through the cundy the return air splits; some travels outbye via the old faceline, whilst the majority returns through the old roadway.

The redistribution of pressure in the district allows an appreciable decrease in pressure differences across the gob, resulting in a large decrease in gob leakage. Control of flow in the leakage intake is maintained by regulators sited at its inbye and outbye ends. These regulators are carefully adjusted so that the airflow through each is the same. A neutral point of ventilation is maintained between main and leakage intakes at some point between the regulators. Outbye of the neutral point there is leakage flow through the gob from the main intake to the leakage intake; inbye of the neutral point the reverse takes place (Figure 5.5). Inbye of the cundy, leakage is from the main intake to the leakage intake, the volume increasing as the distance between the cundy and the faceline increases. It is therefore necessary, to periodically establish a new cundy and seal the old one. Maximum benefit is derived when a cundy is established prior to the cessation of work on the adjacent face and awaits interception. From a practical point of view, it is also advantageous to drive the cundy from a roadway in which coal transportation is available, although the length of time during which the cundy stands prior to use may result in a deterioration in conditions.

While the cundy system controls most of the leakage flow through the gob of a longwall advancing face, it does not significantly affect the total amount of methane emitted. It does affect the paths of methane migration and


25
MAIN INTAKE
25
22
3
12
FACE
7
R
LEAKAGE INTAKE
10
R
7
CUNDY
MAIN RETURN
32
R
LEAKAGE INTAKE
MAIN INTAKE
PRESSURE
PRESSURE DIAGRAM
R
MAIN RETURN

LEGEND

DIRECTION OF MAIN VENTILATION ⟶

DIRECTION OF LEAKAGE VENTILATION - - - ->

TYPICAL AIRFLOW RATES IN m³/s ⑤

CUNDY VENTILATION LAYOUT

FIGURE 5.5

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N. T. S.

procedures must be implemented to control that gas which previously migrated through the gob into the return. This is achieved through dilution using the larger amount of air circulating the district, and by increased application of cross-measure borehole drainage techniques.

During investigation of Lingan Mine (Dames & Moore, H.G. Stephenson, 1977) typical airflows measured in the cundy ventilation system were as follows:

a) 25 m^3/s in the main intake at the outbye end and at a point opposite the intersection of the cundy with the return
b) 22 m^3/s entered the wall, of which 10 m^3/s remained on the face
c) 12 m^3 was near-face gob leakage
d) 7 m^3/s at the inbye and outbye ends of the leakage intake
e) a flow of 32 m^3/s left the district through the cundy.

The cundy system was strongly recommended in the Report of the Commission of Inquiry into the 1979 explosion at No. 26 Colliery, Nova Scotia (Elfstrom, 1980), as a means of reducing leakage through gobs. Figure 5.5 shows the layout and pressure distribution of the system described.

The advantages of the cundy system of ventilation, as applied to the reduction of pressure differential across the gob, are:

a) pressure balance achieved across the major portion of the gob, restricting air loss to that part of the gob between the faceline and the cundy.
b) increased airflow available at intake brushing (up to 90 percent improvement on conventional method), increased face air velocity, improved methane dilution, reduced risk of methane ignition, methane fringe in gob pushed further away from the face.
c) material road in fresh air up to the cundy

The disadvantages of this method of ventilation are:

a) return air courses through old roadways and faceline which may be crushed and difficult to examine and maintain

b) careful control over the regulator apertures is necessary to maintain satisfactory ventilation pressure differences across the gob; problems may result from gob leakage flows entering the main intake, or leakage flow causing high methane concentrations at the inbye regulator to present a serious hazard near to the face when the cundy is newly established
c) passage of material cars through the inbye regulator badly affects balance if any door of the regulator is left open for any length of time
d) the quantity of air entering the main intake is reduced by that quantity entering the leakage intake
e) there is a continuous need to provide new cundies as the basic leakage problem will always be present through the gob inbye the cundy; the distance between the face and the cundy must, therefore, not be excessive
f) the methane/air mixture contained in the gob is affected by changes in barometric pressure; rapidly falling pressure causes this gas body to expand and the resulting flow into the main intake may be sufficient to increase methane concentration to levels where face operations must cease
g) methane concentrations in the adjacent receiving district may reach unacceptable levels (Bowers, 1979)
h) the ventilation pressure difference between the roadways of the adjacent old face causes leakage flow through the gob which can promote spontaneous heating
i) in the event that sealing off should be necessary, an additional two seals, at least, are required
j) the adjacent district cannot be sealed off until the producing district is finished
k) when simultaneous operation of adjacent faces is planned, the application of cundy ventilation to the second face is difficult. Such a circumstance can be provided for by (i) maintaining an extended cundy through the gob of the first face and isolating the ventilation current using two air crossings in which case new leakage paths could present severe problems, (ii) driving a return ventilation roadway through the coal pillar which separates the two faces, or (iii) providing the second face with dual intakes which are separated by a

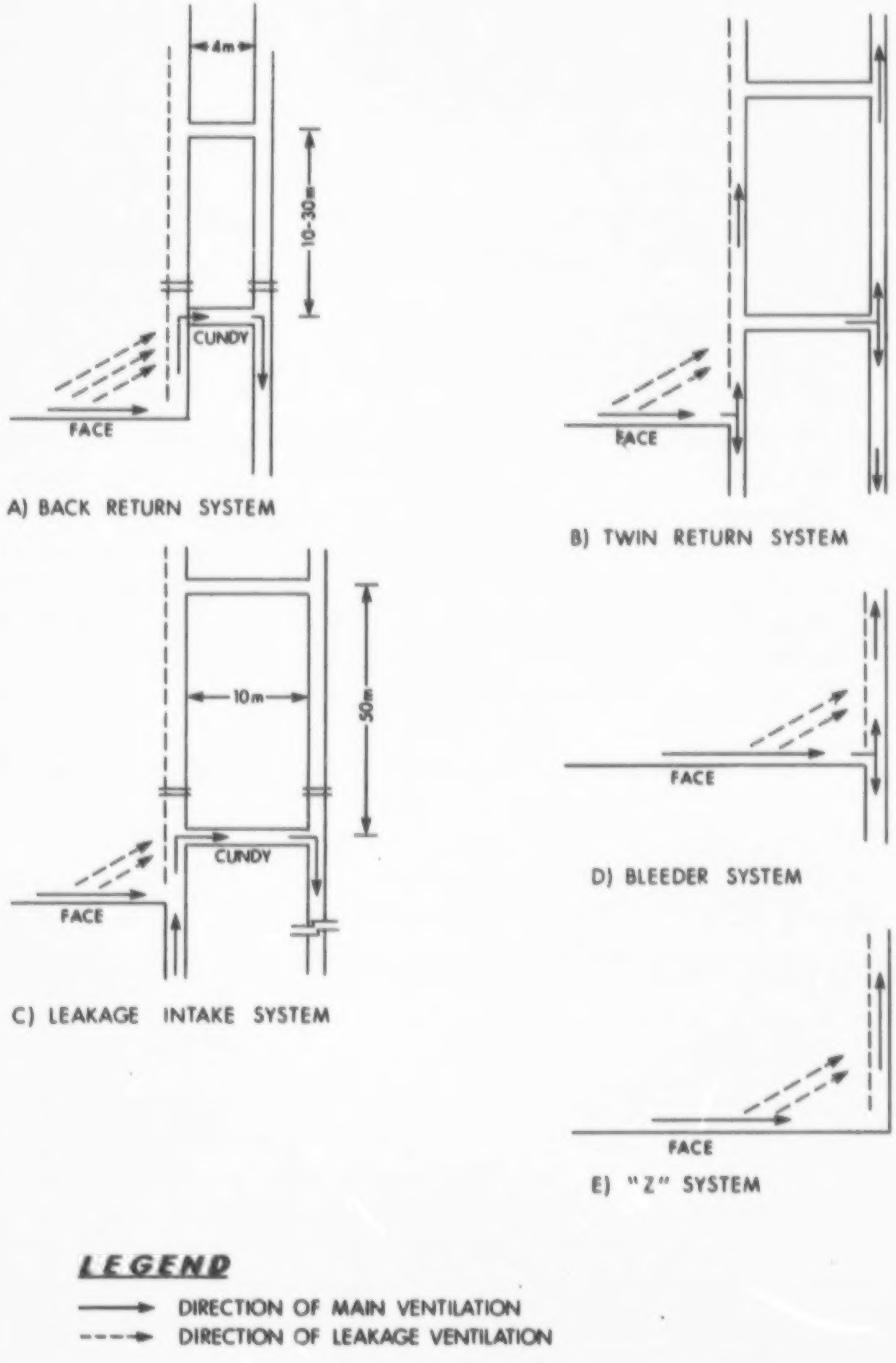
coal pillar; using the roadway protected by coal ribsides as the main intake, and periodically abandoning the second roadway means that the need for a cundy would be eliminated because the coal pillar would effectively prevent leakage

5.8.3 Uses in Longwall Retreating Faces

Longwall retreating faces are usually ventilated by single intake and return roadways with no controlled means of ventilating the gob. The momentum of the air as it enters the face, and the ventilation pressure drop through the face, result in positive ventilation in the gob behind the face supports to a distance which decreases towards the return end of the faceline. The methane fringe, therefore, may be close behind the supports at the face return end, sometimes resulting in a gas flow from the gob over conveyor return end drives and the power loader if it is in that vicinity. This particular methane problem associated with longwall retreat mining is exacerbated on an inclined face, and may be severe on a face retreating to the rise, due to the natural buoyancy of methane. Methane must be prevented from migrating towards this location where potential sources of ignition are present (Thorp, 1970). A variety of methods, some utilizing cundies, can be employed to obtain positive ventilation of a face end gob area and direct the flow away from the face (Massey, 1977). Four of these systems are described briefly below.

(a) Back Return Systems

In this layout, the power loader does not cut through into the return roadway; when it arrives at the face end, a small coal pillar is left (Figure 5.6A). A cundy is driven through this pillar at intervals of 10 to 30 m behind the faceline. The ventilation is coursed from the face return end back to the cundy, through which it passes into the return roadway. The ventilation pressure distribution promotes gob leakage flow away from the face return end area back towards the cundy. The return corner of the face is, therefore, free of contamination caused by methane migration from the gob.


4m
10-30m
CUNDY
FACE
FACE
10m
50m
CUNDY
FACE
FACE
FACE
LEGEND
DIRECTION OF MAIN VENTILATION
DIRECTION OF LEAKAGE VENTILATION

A) BACK RETURN SYSTEM

B) TWIN RETURN SYSTEM

C) LEAKAGE INTAKE SYSTEM

D) BLEEDER SYSTEM

E) "Z" SYSTEM

LONGWALL RETREATING FACE END VENTILATION SYSTEMS

FIGURE 5.6

A similar result may be achieved by the use of twin return roadways which are periodically interconnected by a cundy (Figure 5.6B). The need to develop by a double-entry system may have advantages or disadvantages depending on the prevailing mining conditions, statutory requirements, and costs.

(b) Leakage Intake Systems

Hardman (1976) reported on the application of a leakage intake roadway replacing the conventional return, in longwall retreat mining. After the two airstreams converge at the return end of the face, the air is coursed back along the gob, and then passed through a cundy into the roadway of an adjacent abandoned face. Return air travels outbye through the old roadway. The roadway inbye, to the old faceline, is sealed off (Figure 5.6C).

In the case reported by Hardman, at Holditch Colliery in the United Kingdom, 15 m^3/s entered the main intake and 5 m^3/s was the regulated airflow in the leakage intake. The total flow of 20 m^3/s returned through the adjacent receiving face via the short gob return roadway and the cundy. Throughout the life of the face, the methane concentration at the outbye end of the return averaged 0.7 percent, and the methane drainage capture ranged from 60 to 65 percent, representing a total methane emission rate of 0.4 m^3/s, which was slightly greater than that forecast.

Strata conditions and the method of support of the back return limited its practical length to 50 m behind the face. The pressure drop between the gob edge at the top of the face and the extreme end of the back return kept the methane fringe back from the face and return end conveyor drive. Methane concentrations of the order of 0.3 to 0.6 percent were normal in the back return. Care was taken to build good stoppings in the back return and in the main return as new cundies were formed through the coal pillar. The back return was supported by a double row of cribs on the gobside and a 10 m coal pillar separated it from the adjacent old roadway. Although it was found from experience that the roadway could be maintained for 75 m back from the face, severe ventilation restrictions occurred due to reduced cross-sectional areas. Floor heave affected the back return, but not the leakage intake, demonstrating the effectiveness of the yielding pillar. The maximum 50 m interval between cundies was the optimum compromise between an acceptable ventilation resistance without drivage of an excessive number of cundies. Various cundy sizes were investigated and it was concluded that, ideally, height should exceed width, and dimensions of 3 m by 2 m were used.

(c) Bleeder Systems

In addition to maintaining the two roadways associated with conventional longwall retreating, the bleeder system incorporates a second return back through the gob (Figure 5.6D). Regulating the conventional return roadway causes a controlled airflow to course the gob roadway, and the ventilation pressure differential created near the face return end causes methane migration away from the face. Application of a bleeder system is limited to mining conditions in which it is possible to maintain adequate ventilation through, or adjacent to, a gob roadway of considerable length.

(d) "Z" Systems

In the "Z" system, the return roadway of the conventional retreating face is eliminated and all air leaves via a gob return roadway (Figure 5.6E). The problem of maintaining the single return through an extensive gob severely restricts the application of the system.

5.9 HYDRAULIC MINING

Hydraulic mining is employed at the B.C. Coal operation in Southeastern British Columbia, the only hydraulic coal mine operating in Canada. The mine is producing from a 15 m thick seam of soft coal in the Kootenay formation, where the strata dips at approximately 30°. This is the thickest seam being extracted by underground methods anywhere in Canada. Underground operations in a thick, soft, dipping seam are one of the most difficult technical problems facing the mining engineer. Obtaining an acceptable recovery of the reserve, in a safe efficient manner, is difficult whichever mining method is used.

The hydraulic mine at Sparwood has been successful in obtaining a recovery of the reserve appreciably higher than a room and pillar operation in similar mining conditions, whilst maintaining a high standard of safety and productivity.

The hydraulic mine utilizes a monitor to direct a jet of high pressure water at the coal face. Coal broken from the face is carried by the water into an open flume which transports the coal/water slurry away from the extraction area.

In common with other forms of mining, the hydraulic system utilizes a development phase followed by the extraction phase. Main and secondary entries divide the mining area into coal blocks to be exploited. Each block is then developed into production panels by driving a subheading down which the coal and water can be flumed. The subheading is driven to the rise by a continuous miner loading into a water-fed flume. At some future time the mechanical continuous miner could be replaced by monitor face cutting equipment in development headings.

Once the subheading has advanced to its planned distance, the heading equipment is replaced by a monitor. Solid coal on one or both sides of the subheading is broken off by the water jet and washed into the flume. The monitor is retreated back down the subheading in suitable increments until the panel is mined out. It is common practice to maintain three panels:

a) one with an operating monitor
b) one with a monitor on stand-by
c) one with a monitor being pulled back.

In order to maintain monitor places it is usual to have at least two and possibly three panel subheadings under development at one time. The levels of production obtained from the monitor are significantly affected by the seam thickness. Increases in production are obtained by increasing the number of operating monitors and increasing the number of development headings to maintain them. (Parkes and Grimley, 1975)

Hydraulic mines fall into two categories:

- those mining areas "above drainage" where the coal can be flumed directly to the surface via a rock tunnel or outcrop entrance and,
- those mining from lower levels where coal is flumed to a low point in the mining area and raised to the surface by mechanical means such as conveying, pumping or a combination of methods.

Ventilation systems utilized in the lower levels of the mine below drainage may be different from those employed in extraction areas close to the outcrop.

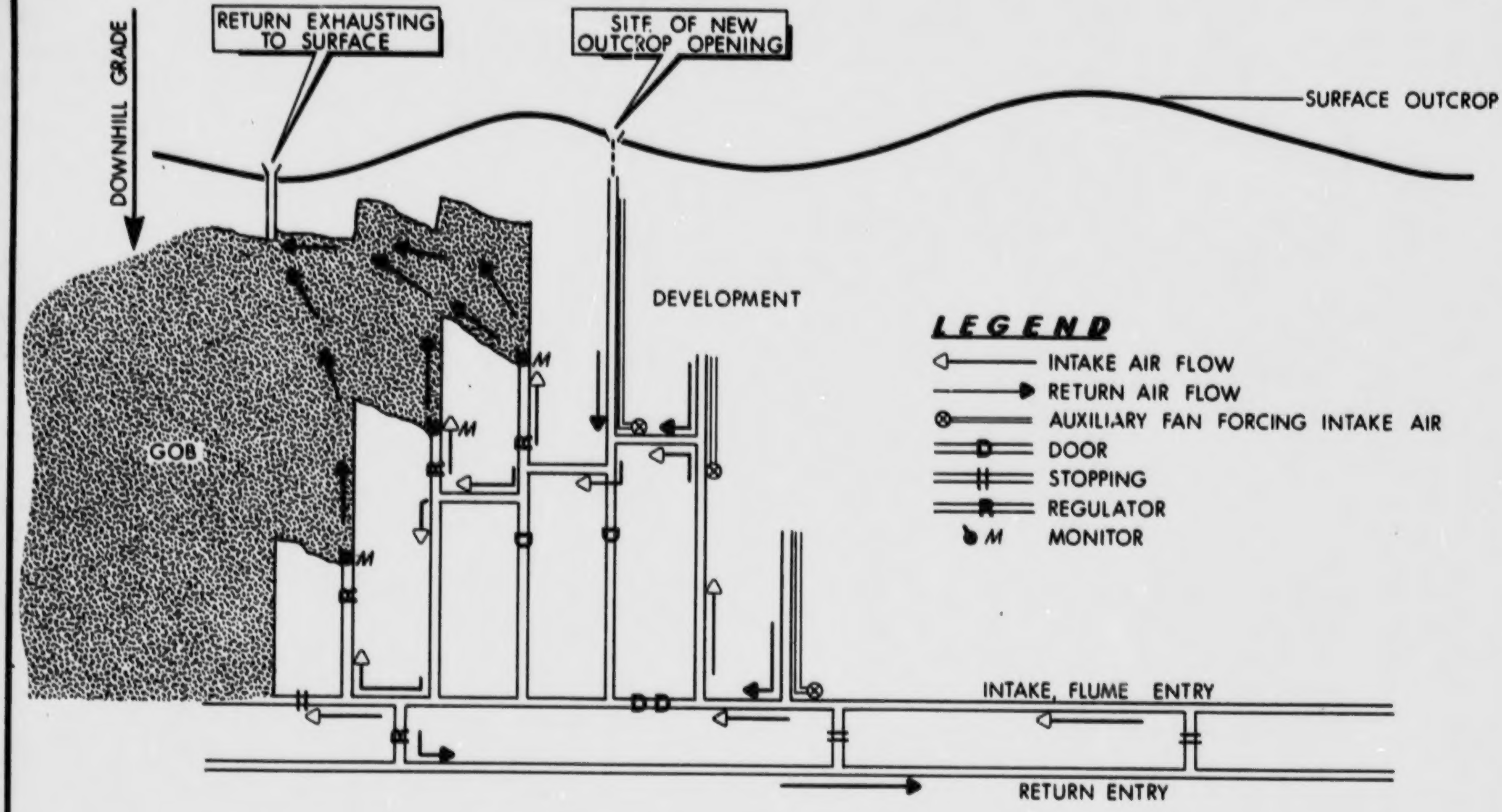
5.9.1 Ventilation

The ventilation of development headings in hydraulic mines, as currently practiced, is by conventional auxiliary fans and ducting. There are no special problems attached to this except that long single headings must be driven to form sub-levels. The length of these sub-levels is critical to the efficiency of the hydraulic operations and, within reason, the longer they are the higher the reserve recovery and productivity levels. However the problems of ventilating these headings to dilute dust and methane can be a limiting factor in the length of the pillar to be developed. This is a similar problem to that encountered in retreating longwall systems using single entries.

In a thick seam where headings are driven close to the floor, support is provided by steel arches. Advancing under a coal roof to make room for the next arch sometimes leads to overbreak of the coal above the level of the arch crowns. This occurs particularly in soft coal or where immediate pressure is exerted by the surrounding strata. The result is a cavity over the arch where methane can collect even though the cavity is supported by timber. Although layering and accumulations of methane over the tops of arches is related to ventilation, remedial action is most likely to be related to strata control and is not discussed further here. This problem is encountered in any mining operation where headings are developed in a soft thick seam and friable strata is left in the roof.

The retreating monitor panels are ventilated by passing intake air up the panel to the monitor and allowing it to return through the mined out gob. In Canada, the areas mined to date have been immediately below the outcrop. By making occasional outlets to the surface along the outcrop, the air returning through the gobs is exhausted to the atmosphere. Where an outcrop is not accessible, the return air from the gobs must enter a return airway sometimes referred to as a "bleeder road" or alternatively, as a "false outcrop". The system described above is known as an "open gob" system of ventilation (See Figures 5.7 and 5.8). Success depends on the gob areas of the retreating monitor panels remaining open long enough to offer a return air route during the life of the panel. In thick seams, the outcrop outlets and the "bleeder returns" are best driven at, or close to, the top of the seam. A return near the bottom of the seam can very easily be blocked off by caved roof debris and the air flow prevented or severely restricted (See Figure 5.8).

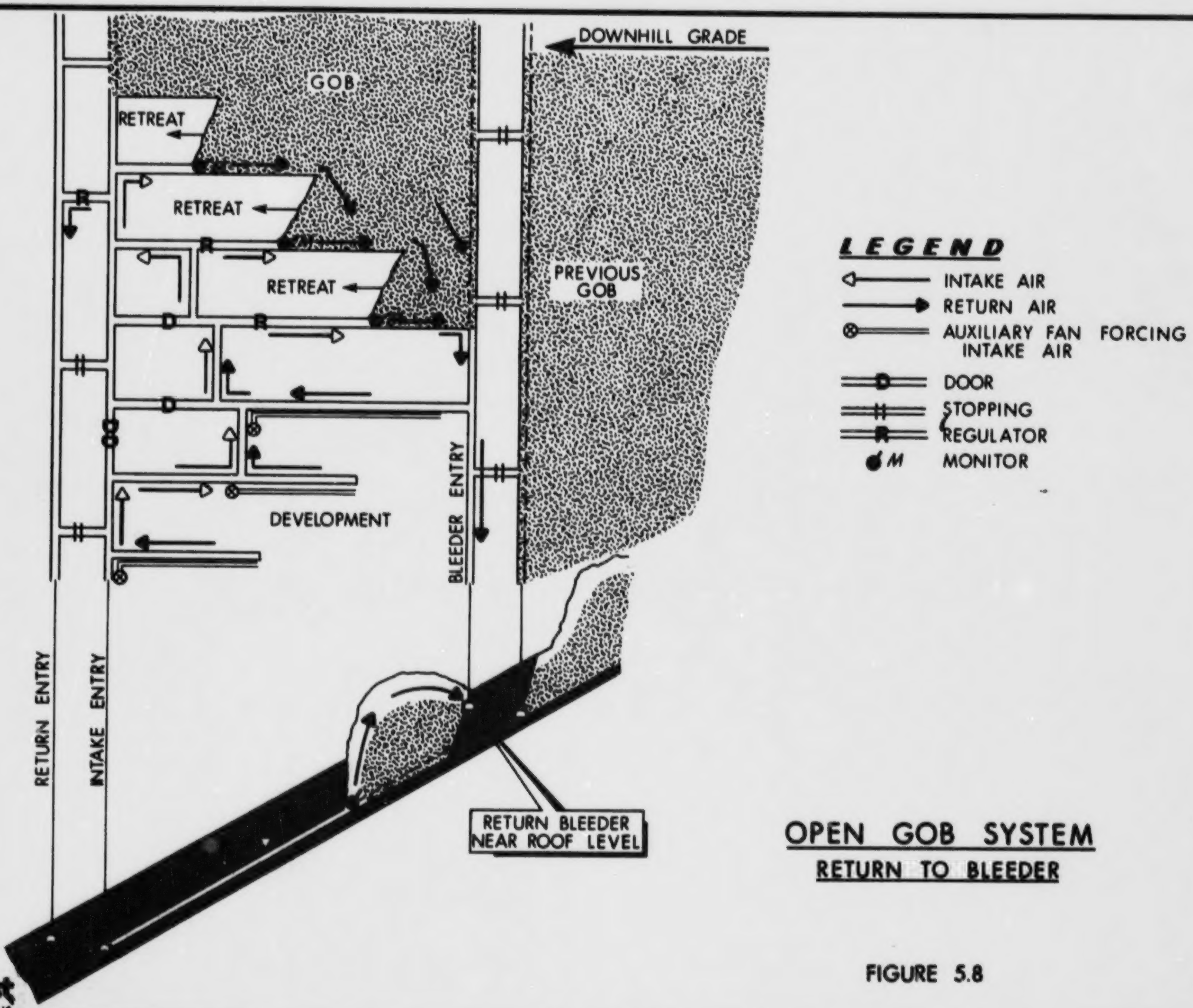
A "closed gob" system has not yet been practiced with the hydraulic mining method in Canada. The "closed gob" system would require the air returning from the monitors to pass through ventilation ducts, using either the ventilating pressure created by the main fan, or auxiliary fans and tubes. Alternatively, an adjacent roadway or some combination of the above mentioned methods, might be used. Illustrations of possible ventilation circuits are given (Figures 5.9 and Figure 5.10). Modifications and variations of these circuits will be necessary to meet a variety of circumstances.


DOWNHILL GRADE
RETURN EXHAUSTING TO SURFACE
SITE OF NEW OUTCROP OPENING
SURFACE OUTCROP
DEVELOPMENT
GOB
M
LEGEND
INTAKE AIR FLOW
RETURN AIR FLOW
AUXILIARY FAN FORCING INTAKE AIR
DOOR
STOPPING
REGULATOR
MONITOR
INTAKE, FLUME ENTRY
RETURN ENTRY

OPEN GOB SYSTEM

RETURN TO OUTCROP

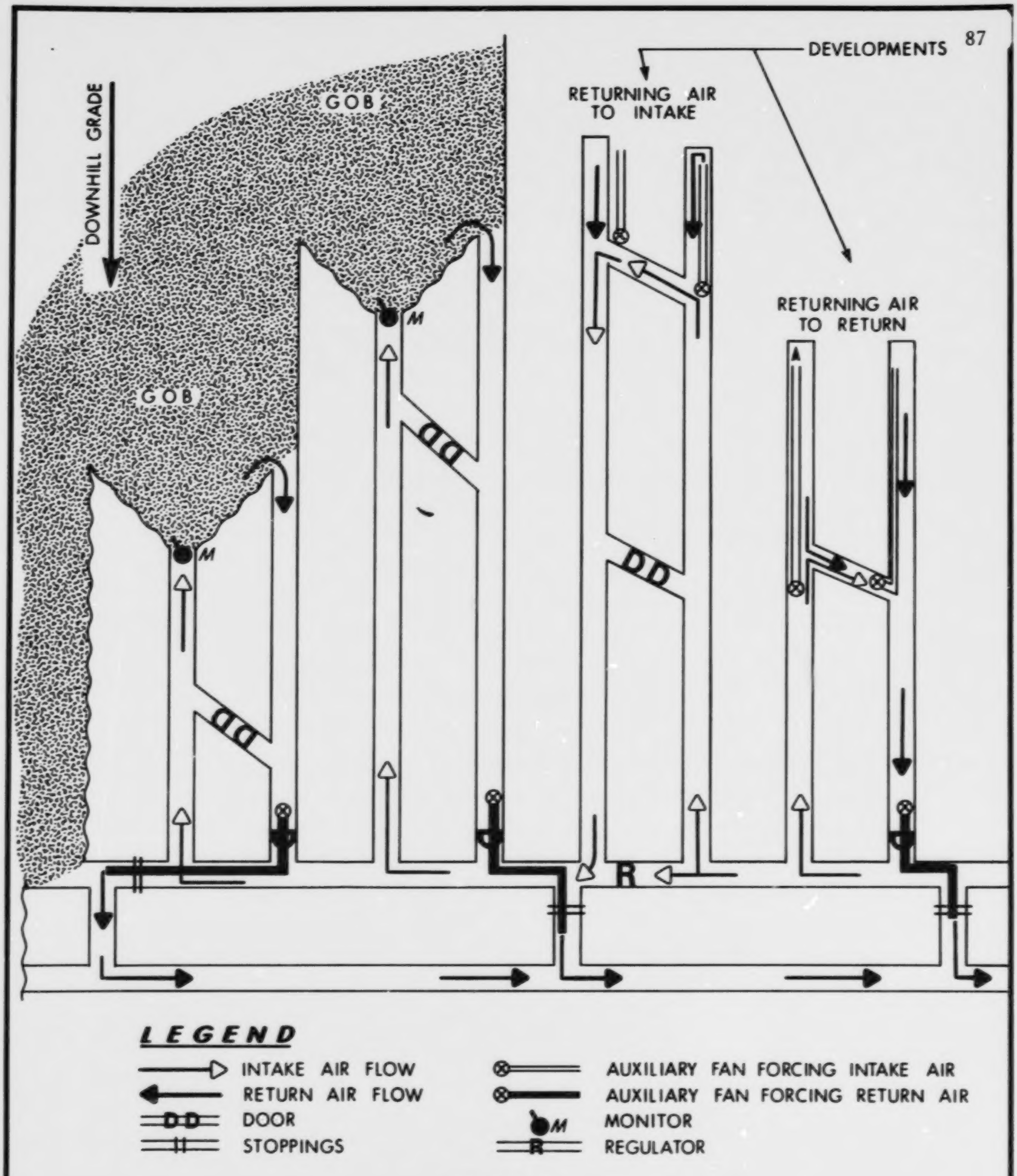
FIGURE 5.7


DOWNHILL GRADE
GOB
PREVIOUS GOB
RETREAT
RETREAT
RETREAT
DEVELOPMENT
RETURN ENTRY
INTAKE ENTRY
BLEEDER ENTRY
RETURN BLEEDER NEAR ROOF LEVEL
R
R
R
D
D
D
LEGEND
INTAKE AIR
RETURN AIR
AUXILIARY FAN FORCING INTAKE AIR
D DOOR
STOPPING
R REGULATOR
M MONITOR

OPEN GOB SYSTEM
RETURN TO BLEEDER

FIGURE 5.8

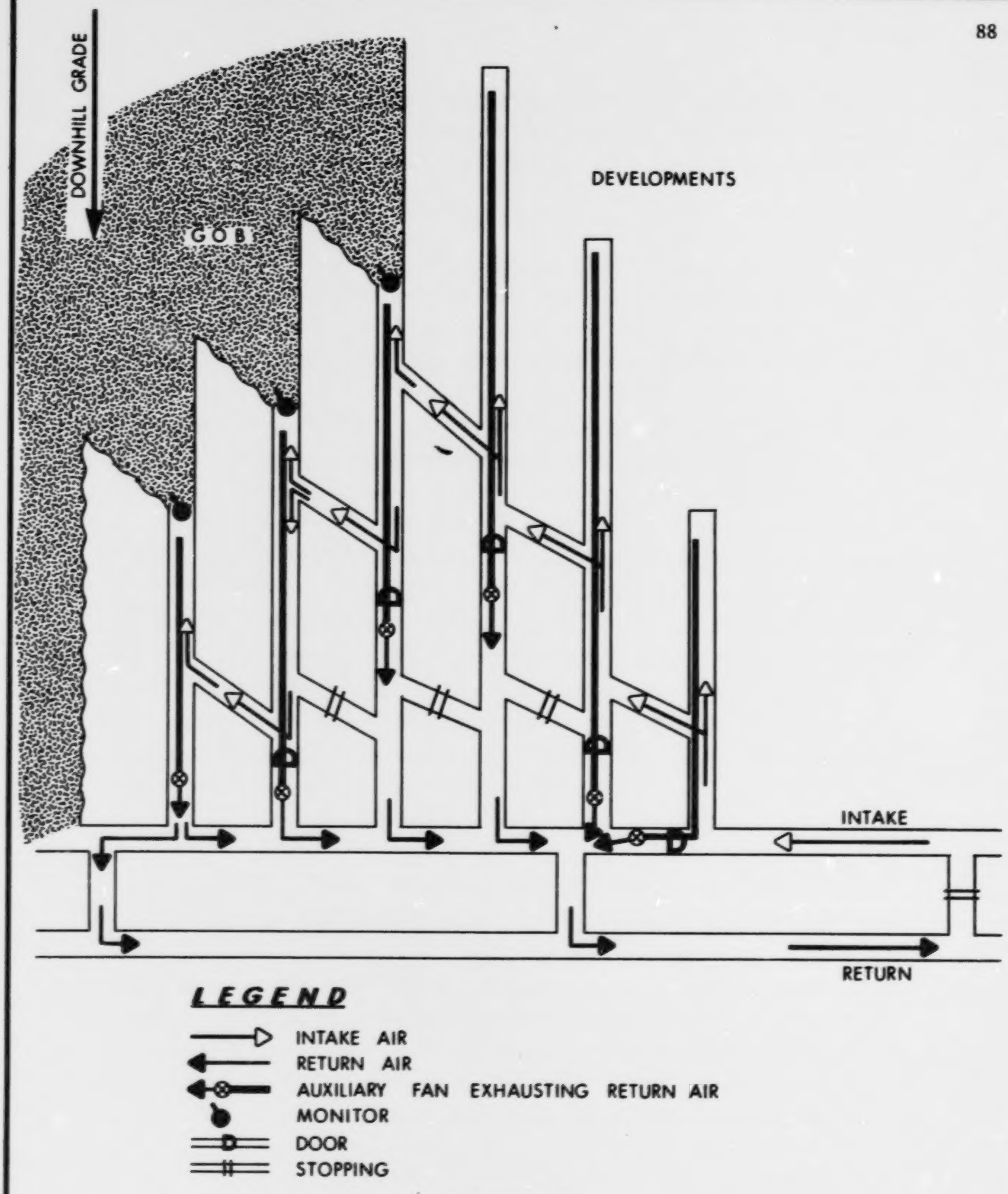
NorWest Resource Consultants Ltd


DEVELOPMENTS
RETURNING AIR TO INTAKE
RETURNING AIR TO RETURN
DOWNHILL GRADE
GOB
GOB
M
M
DD
DD
DD
R
LEGEND
INTAKE AIR FLOW
RETURN AIR FLOW
DD DOOR
STOPPINGS
AUXILIARY FAN FORCING INTAKE AIR
AUXILIARY FAN FORCING RETURN AIR
M MONITOR
R REGULATOR

SCHEMATIC
CLOSED GOB VENTILATION
(USING MONITOR ON TWO SIDES)

FIGURE 5.9

NorWest
Resource Consultants Ltd


DOWNHILL GRADE
GOB
DEVELOPMENTS
INTAKE
RETURN
LEGEND
INTAKE AIR
RETURN AIR
AUXILIARY FAN EXHAUSTING RETURN AIR
MONITOR
DOOR
STOPPING

SCHEMATIC
CLOSED GOB VENTILATION
(USING MONITOR ON ONE SIDE)

FIGURE 5.10

NorWest Resource Consultants Ltd

In comparing the open and closed gob systems and the various methods of coursing the air, it must be emphasized that it is not a question of one method or air circuit being better than another; it is a question of arriving at a ventilation system which adequately meets statutory safety requirements and provides a reasonable working environment that complies with the established safety levels for dust and noxious and flammable gases.

Some of the factors which will play a part in evolving a satisfactory ventilation system are briefly noted below:

a) The situation in the gob areas, and particularly their suitability for use as return airways.
b) The extent to which return air from development headings is contaminated with dust or methane and is, therefore, unsuitable for use in an adjacent separate face.
c) The availability of sufficient ventilating air to allow auxiliary fan systems to operate without danger of recirculation.
d) The susceptibility of the coal to spontaneous heating and the liklihood of methane collecting in the gob areas. (More will be said about spontaneous heating and hydraulic mining with open or closed gob systems in the following Section 5.9.2.)
e) The ability of this ventilation system to clear methane from the pillar face in very thick coal when high emission rates are encountered.

f) The depth of cover and/or proximity to the outcrop. If breaks and openings to the surface are formed as the coal is extracted, then it may prove impossible to maintain a "closed gob".

5.9.2 Spontaneous Heating

Investigations have shown that the heat released by moist oxidation of coal is greater than for dry oxidation, and that heat is released due to the adsorption of moisture by the coal. (Hodges and Acherjee, 1966).

Experience in the hydraulic mine in southeastern B.C. is that most heatings occurred during mid-winter when the mine air was dry. Under these conditions, one might expect that increased evaporation would bring about an increased cooling effect. Alternatively, during these cold months, the natural ventilating pressure effect in a steep seam will be assisting the forcing main fan. Air finding its way through the gob and then out through surface breaks may, under this extra pressure, penetrate deeper into the old gobs. Under these conditions, any incipient heating which was previously in a state of thermal equilibrium could soon become active. It is possible that a similar change in ventilating conditions had occurred on a previous occasion when supply cars traversing a roadway were restricting the cross-section like "a moving regulator". It was surmised that this caused intermittent air flows through adjacent, unused, roadways so that a heating developed.

A discussion on spontaneous heating in underground hydraulic coal mines cannot ignore the necessity of comparing the "open gob" and "closed gob" alternatives. In longwall mining, advancing faces tend to utilize a gob which is partially open and ventilated by leakage from intake to return. A retreating face is virtually a closed gob situation with very minimal leakage passing through the mined out areas. In hydraulic mining of a thick, dipping seam a similar choice has to be made between the use of "open" and "closed" gobs.

The case for the open gob system, simply stated, is as follows:

a) Air flow through the gob will carry away heat generated by oxidation of the coal; the possibility of a heating is therefore reduced.
b) With the gob ventilated it is likely that concentrations of methane will be diluted sufficiently to avoid the creation of an explosive mixture.

For the closed gob, the case is:

a) Circulation of return air through an uncontrolled, non-maintained roadway (the open gob) is avoided.
b) If air is not directed through the gob there will be an insufficient supply of oxygen for spontaneous heating to develop.
c) Although heatings are less likely to occur any explosive concentrations of methane in the unventilated gob would be liable to ignite if a heating did develop.

In Canada, only the first system has been practiced to date in hydraulic mining. There have been heatings with minor losses of production and reserves, but no loss of life and no ignitions of methane.

Although the closed gob appears to offer a practical alternative, in many cases it will be almost impossible to prevent air from finding its way into the gob. This is particularly true where the operation is in a thick seam, and especially where it is close to an outcrop. If a coal is liable to spontaneous heating, the first objective should be to leave as little coal as possible remaining in the gobs. With hydraulic mining methods, most unrecovered coal is left crushed and waterlogged; when free water drains out of the broken coal, air enters, regardless of whether it is an open or a closed gob. The second objective is to detect a heating at the earliest possible time. Constant monitoring of the CO content of the return air is the most common form of early detection. When an open gob system is exhausting to the outcrop, effective monitoring of the return air is often difficult. When a bleeder road return is used, monitoring for CO is more easily accomplished.

5.10 METHANE DRAINAGE

When methane emission rates are high and large tonnages of coal are being produced from a small area, dilution of methane to acceptable levels by general ventilation becomes difficult or even impossible. There is a practical limit to the quantity of air that can be passed through a working face, whether it is a longwall or a single entry development. There is a point, dictated by both economic and technical considerations, when good ventilation may have to be supplemented by other means of methane control. Methane drainage is the most common and effective answer to this problem.

Methane drainage practices have, since the late 1940's, become increasingly popular in Europe. Very sophisticated and efficient systems can now be engineered with a high degree of predictability as to performance. In North America, adoption of methane drainage commenced much later, because the dominant system of mining, room and pillar operations, did not result in such locally concentrated emissions of methane, and because application of drainage techniques has always been more difficult in room and pillar operations. As longwall mining became more popular in North America, use of methane drainage systems became common, though the methods used often differed from those favoured in Europe. In Western Canada only very limited, small scale, attempts at methane drainage have been made.

The mines operated by the Cape Breton Development Corporation are an exception to the comments made above. These mines have tended to follow European methane drainage practice largely because the mining method used, advancing longwall, is the same as in Europe, particularly the United Kingdom.

Although methane drainage is not, strictly speaking, a ventilation strategy, the techiques are inseparable. In this report, frequent reference will be made to methane drainage and therefore a short introductory description of the techniques available is essential. A detailed description of the subject is contained in a report by Dames & Moore and H.G. Stephenson (1977a).

All methane drainage techniques are based on the principle of draining a highly concentrated methane/air mixture from the strata and transporting it to a location where it can be vented safely. Drainage systems have usually been employed in conjunction with, or prior to, extraction of the coal. It is only applications of drainage where safety of the mine is a primary consideration, that are of interest here.

In almost all drainage systems, suction pumps are employed to draw the mixture into pipe ranges for transportation to the vent point. In some cases the gas is piped to a location on the surface where it may be vented or utilized for heating or power generation.

In Europe, methane drainage is almost always applied to the strata or the gob in the vicinity of the operating face. Here, coal extraction has disturbed the strata and gas flows easily, through cavities or fractures, to the point where suction is applied. In such cases, the proportion of methane captured by the drainage system ranges from 10 to 80%. In North America, however, drainage of methane in advance of mining has been quite successful in highly permeable coal seams. These are frequently encountered in North America, though relatively uncommon in Europe.

Some of the methane drainage systems used are described briefly below.

5.10.1 Cross-Measure Boreholes

Worldwide, this is the most widely applied system. Boreholes are drilled across the strata adjacent to the seam being mined. The boreholes are used to draw methane from the strata surrounding the gob, and indirectly from the gob itself. The gas/air mixture is drawn into a pipe range by suction. In almost all cases holes are drilled at right angles to the roadway and parallel to the face. Hole lengths range from 20 to 90 m. Average spacings are approximately 9 m in the roof and 22 m in the floor. Drainage is most commonly practiced from the return airway, though in some cases boreholes may also be drilled from the intake. Standpipes are inserted and sealed in each hole, and flexible hoses connect the standpipe to the pipe range.

The cross-measure borehole system is applied with many variations. One of the most unusual was at Holditch Colliery. (Dames & Moore, H.G. Stephenson, 1977a). Very short holes (2 - 2.5 m long) are drilled in clusters into the roof and floor of the seam, and connected into the drainage system. This technique allowed the operator to drive single entries of up to 1200 m in a fairly gassy seam. As a result, ventilation of these very long single entries was achieved using standard equipment. The use of single entries, in turn, increased the recovery from retreat longwall mining and reduced strata subsidence problems.

Some of the most important parameters for effective utilization of the cross measure borehole system are:

a) Boreholes must intersect the gas-bearing strata.
b) Boreholes must be maintained as close to the face as possible.
c) Standpipes must be of adequate length and good seals must be maintained.
d) Sufficient suction must be available at the boreholes.
e) The pump and pipe systems must have adequate capacity.

In Canada the cross measure borehole system is applied extensively at No. 26 Colliery and Lingan Mine in Cape Breton (See Section 8). If the system was not in use at these mines, it is likely that they could only operate at a significantly reduced level of production.

5.10.2 Superjacent Headings

A gas drainage heading (or headings) is driven in strata above the seam to be mined. The heading may be in a thin subeconomic seam, in the gob of a mined out seam, or in rock. If existing roadways are accessible in an upper seam which has already been mined, they may be utilized. Clusters of holes are drilled from the superadjacent heading, and intersect the seam to be mined. The gas is discharged through a pipe range as with other systems.

5.10.3 Pack Cavities

Pack cavities are utilized where solid stowing is employed behind longwalls. Cavities are left in the stowed area and a pipe, built into a pack at the rise side of the face, is used to drain the cavity.

5.10.4 Sealed Areas

This method is used to control emission of gas from sealed gob areas. Drainage pipes are installed through pressure chambers which seal off the area. The methane/air mixture within the sealed gob is drawn off as required tc prevent uncontrolled leakage into the workings. Leakage occurs most often when the barometric pressure is falling.

5.10.5 In-Seam Holes

This method is useful when a seam of high permeability is to be mined. In this case it can be used effectively during the development stage. As the name implies, the technique involves drilling holes in the coal seam and applying suction in the usual way. In some instances very long drainage holes have been utilized but in these cases it is obviously important that the holes be kept in the seam when they are being drilled.

5.10.6 Drainage From Mine Shafts

In the United States, shafts have been used to place a drilling machine at a point in the seam well in advance of mining. By drilling and draining a pattern of holes, very large quantities of gas have been drained with a significant improvement in ventilating conditions when the faces eventually reach the area affected.

5.10.7 Isolation of Panels

A panel of coal is blocked out and allowed to remain unmined for a period of time. Providing seam permeability is reasonably high, the quantity of gas which drains away during this period may be sufficient to significantly reduce the quantity of methane emitted when mining does commence.

All the methods listed above in 5.10.1 to 5.10.7 utilize underground access to drain methane. There are, however, methods which do not require underground access; they are described briefly below.

5.10.8 Holes Into Virgin Coal

Boreholes drilled from the surface to intersect the coal may produce significant quantities of methane, providing strata permeability and reservoir presure are sufficiently high. A very significant increase in gas production is obtained if hydraulic stimulation is employed. Drainage holes have been drilled from the surface, and deflected along the plane of the coal seam, before suction was applied to draw off the methane.

5.10.9 Holes Into Gob Areas

Holes are drilled from the surface into gob areas and gas is drained off, usually by suction. This method has been used very effectively in the United States. Two attempts have been made in Canada, at Canmore and Cadomin, Alberta, but neither produced sufficient methane to signficantly improve conditions underground (Dames & Moore, H.G. Stephenson, 1977a)

5.11 VENTILATION ON COAL HAULAGE ROADS

In the U.S. Regulations restrict air velocities to a maximum of 1.27 m/s in main haulage roads where coal cars and locomotives are used. Canadian practice is less specific, generally allowing the Mines Inspector to use his discretionary powers to restrict velocities if there is a dust problem.

However, as mines become more extensive and production capacities increase, the problem of dust in haulage roads will probably become more severe. This is particularly true for areas like Nova Scotia where future plans for extending deep submarine mines envisage very long access tunnels in rock. These tunnels will be required to handle large air volumes at high velocities as well as high speed coal transportation. In order to avoid dust problems, design and planning of the mines should pay special attention to:

a) having the main ventilation by-passing the loading stations
b) improved dust control at loaders and transfer points, including the use of vacuum hoods, chemical additives and foam for dust suppression
c) the use of chemicals to spray and seal the loaded coal, or a covered top on the coal car.

6. VENTILATION APPLIANCES AND LEAKAGES

6.1 VENTILATION STOPPINGS

In Canadian coal mines ventilation stoppings are constructed of concrete blocks, wood and/or plastic brattice. There is general agreement that methods of construction are not wholly satisfactory.

The ideal ventilation stopping would be:

a) Easily and rapidly constructed;
b) Built with materials which can be easily transported and stored;
c) Constructed of fire proof material;
d) Resistant to damage from strata movement;
e) Easily repaired when damaged;
f) Highly resistant to the passage of air; and
g) Easily adapted to uneven strata around the perimeter of underground drivages.

Ventilation stoppings now in use meet some of the above requirements. The method of construction and the care taken in each case depends on the importance attached to the particular stopping and the practical difficulties encountered by the mine operator.

In most mines, due to strata movement such as crushing of pillars and floor lift, stoppings have to be repaired frequently and are a source of significant ventilation leakage. The problem is particularly acute in those areas of the mine where total caving is practiced i.e. depillaring areas or longwall panels. The difficulties of erecting and maintaining sound stoppings are greatest in thick seams of friable coal. The demands on a stopping are greatest when high ventilating pressures are necessary and/or when the area inbye the stopping receives barely sufficient air.

In outbye areas, where ground is more stable and pillar extraction is not contemplated for many years, stoppings are generally constructed of cement cinder blocks. This type of stopping is fireproof and can be sealed effectively, into the perimeter of the roadway. Unfortunately, if strata movement does occur, the stopping has little or no resilience and cracks are formed which are difficult or impossible to seal. The practice of incorporating horizontal layers of wood between rows of blocks rarely provides sufficient cushion to prevent cracking.

In areas where strata movement is expected, most Canadian mines utilize stoppings constructed of timber and plastic brattice. Sometimes these stoppings are sprayed with sealant to reduce leakage around the perimeter and through joints. The sealant will also assist in rendering the stopping fire resistant. Such stoppings are favoured for inbye areas because they are easier to repair than concrete block stoppings and the materials are more easily transported.

Some Canadian mine operators are looking at prefabricated metal stoppings. Tien (1981) reports on the use of aluminum stoppings at a mine in Illinois. The stopping is intended to accomodate some movement of the roof and floor and depends partly on the flexibility of aluminum sheet. If a similar stopping were considered in Canada aluminum would have to be replaced by steel to satisfy the mine legislation.

Other prefabricated metal stoppings originating in the United States have been considered by Canadian operators. Very few have been utilized in Canada.

6.2 VENTILATION DOORS

Ventilation doors are fitted into a wall constructed in the same manner as the stoppings already described; they therefore suffer from the same problems of damage due to crushing.

Both wooden and steel doors are used in Canadian coal mines. Generally speaking larger doors, used to permit the passage of vehicles, are most often constructed of steel. They may be fabricated at the mine or purchased complete with frame.

Steel doors are difficult to repair underground when damaged by the passage of vehicles or strata movement. At inbye locations, susceptible to this type of damage, doors made of wood are favoured.

Maintenance of an effective seal at floor level, and where double doors meet, is a problem.

Most operators agreed that they had problems maintaining doors and stoppings in satisfactory condition to minimize leakage. There was little evidence of any moves away from the traditional methods even though many different materials and methods are being applied in other countries, particularly in Britain and the U.S. These include patent sealants, gypsum compounds such as "Hardstem", and fibreglass compounds. It is apparent that the mobile pumps, sprays, applicators, and power supply necessary to apply these new types of stopping, as well as the need to clean the equipment after use, are a deterrent to the operators.

6.3 LEAKAGES

In most Canadian mines, only between 50 and 60% of the air handled by the main fan reaches the operating sections, and not all of this actually reaches the coal face. It must be taken into consideration that some of the intake air is deliberately short circuited to provide sufficient ventilation in areas such as transport roads with diesel powered vehicles, pumping stations, underground silos and bunkers, etc. Nevertheless, ventilation costs at a large mine are such that reduction of leakage can result in a considerable reduction of cost (Pickering, 1982).

The main sources of leakage are identified as follows:

a) Stoppings, doors, regulators and air crossings damaged by ground movement.
b) Ventilation doors damaged by passing vehicles.
c) Conveyors passing through doors.
d) Air leaking around portals and air locks at main fan installations.

e) Leakage through the body of stoppings or doors which in practice cannot be prevented unless a perfect stopping is developed.

Each of these sources are examined briefly in the following paragraphs.

Damage from ground movement has been discussed previously.

Damage from passing vehicles, either by accidently catching the door frame or by using the vehicle to push open the doors, is a common cause of leakage. The use of larger, steel, prefabricated door structures would reduce this problem in some instances. Where vehicles pass regularly, especially where there are high pressures, automatic or semiautomatically operated sets of doors are recommended. McIntyre Mines Ltd. have installed a set of semiautomatically operated doors at the forcing fan portal of their 9G-4 Mine. The doors were purchased as a unit from American Mine Door and are opened and closed by the vehicle drivers operating hydraulic valves. A similar door is in use in the Atlas mine. In this case the doors are on a locomotive haulage road. The doors are opened and closed by a mechanical linkage operated by the weight of the train depressing a rail hinged at one end. The use of these types of devices, particularly at portals, should be encouraged.

Where conveyors pass through ventilation door structures, leakages can be significant, particularly where the ventilating pressure across the structure is high. Air losses of up to 10 m^3/s are not uncommon due to the impossibility of securing a satisfactory seal around the top and bottom belts of the conveyor. In addition, high velocity air flow through the conveyor opening will raise dust off the belt. Attempts to reduce the leakage are usually confined to building a series of door stoppings. Because of the difficulty of obtaining satisfactory seals, and because of the dust problem created, ventilation systems should be planned to avoid the use of these structures as much as possible, especially in high pressure zones.

Air leakage at main fan portals, where pressure differences are greatest, can be high and very difficult to limit unless the portal has been well designed and maintained. The most difficult area to seal is the periphery of a portal where it is built into the strata. The use of grouting and concrete filling behind shuttering is now less common in Western Canada's mines than it was up to the 1950's; possibly because today's mines have a shorter life due to increased mechanization. Portal maintenance is another important factor. Leakages that can develop due to deterioration of portal structures and erosion around the structures, especially in the steep mountain and foothill areas, can lead to severe air losses. One mine was experiencing a loss of 23 m^3/s at an exhausting fan portal due to erosion and deterioration; this portal had originally been well constructed of steel and concrete. When mine air heaters are used with forcing main fans, portal leakages are particularly costly due to the loss of heated air.

7. FIRES AND VENTILATION PRACTICE

7.1 INTRODUCTION

The subject of mine fires and their control was dealt with at some length in a report "Survey and Assessment of the Mine Fire Prevention and Control Practices in Coal Mines" (Dames and Moore, and H.G. Stephenson, 1979). This report will be referred to but duplications will be avoided as much as possible. The objective here is to confine discussion to the interaction of underground fires and ventilation rather than analyze the cause of the fires.

The type of fires found in underground mines fall into two broad categories:

a) Spontaneous heating: where oxidation of organic materials - usually coal - commences at ambient temperatures, generating sufficient heat to result in smouldering of coal and sometimes in open fire;

b) Open Fire: where ignition of flammable materials is caused by some external source of heat such as electrical arcing, frictional sparking or rubbing or some other heat source. The rate of burning is affected by the supply of air both in providing oxygen to the fire and in fanning the flames.

7.2 SPONTANEOUS HEATING

7.2.1 Causes

Listed below are some of the main factors affecting the incidence of spontaneous combustion.

TABLE 7.1
MAIN FACTORS CONTRIBUTING TO SPONTANEOUS COMBUSTION

Chakravorty, Feng, Cochrane (1973)

Intrinsic Factors Which Cannot Be Controlled		Extrinsic Factors Which Can Be Controlled
Coal Properties	Geological Features	Mining Practice
1. Low rank	1. Presence of faults in coal seams	1. High losses of coal in worked out area
2. High moisture	2. Weak and disturbed strata condition	2. Excessive fissuring due to strata movement (stress relief)
3. High pyrite	3. Thick seams with bands of inferior coal	3. Caving to surface under shallow overburden or near outcrop
4. High friability	4. Seams under shallow overburden	4. Mining of thick seam by caving - partial extraction of seams
	5. Multi-seams in close proximity	5. Ventilation anomalies -imbalances, constrictions, large pressure differential, air leakages etc.

Weak strata will usually require the use of longwall mining, leaving a coal roof and selectively mining the better parts of the seam. The unrecovered coal will be left behind in the gob as a spontaneous combustion risk. Sections 5.9 and 10.3 discuss the circumstances of hydraulic mining as they affect spontaneous combustion and need not be repeated.

Although unrecovered, broken coal, either left behind in gobs or crushed off the ribs of coal pillars, is generally considered to be the material which oxidizes, it is possible that other materials can be involved too. For example, burning wood and remnants of timber supports have been noticed at many heating sites. It is not clear whether -

a) the timber itself oxidizes and spontaneously combusts,

b) the timber is simply a material with a low ignition temperature ignited by the heat of oxidizing coal; or whether

c) the presence of wood simply provides air passages enabling oxygen to be supplied to the seat of the heating.

7.2.2 Prevention

Some of the principal steps which can be taken to prevent spontaneous combustion underground are listed below.

a) It is important to recover as much coal as is economically possible and leave as little timber as possible. Where pillars must be left, they should resist crushing and/or be isolated from the air by a compacted gob, or by stoppings, before heatings develop.

b) Air leakage across gobs must be minimized. This is particularly important for advancing longwalls. A distinction should be made between "leakage" across a gob and "open gobs" where a gob is used as an airway (See Section 5.9). Leakage across gobs can be reduced by:

   i) sealing off the roadway sides;
   ii) minimizing the pressure differential across the gobs; and
   iii) maintenance of a high standard of strata control.

If return air is to be circulated through a gob a system which permits the air to be directed through a known, low resistance, path will be preferable to uncontrolled random leakages. Achieving this is often very difficult.

When a retreating longwall system is utilized in a gassy seam, a bleeder system may provide a better environment on the face. However a bleeder system may, in a seam liable to spontaneous combustion, increase the risk of a heating.

7.2.3 Detection

Early detection of spontaneous heating is now commonly done by continuously monitoring for CO by a Tube Bundle system. However, for smaller mines, frequent spot-sampling for CO at regular intervals will probably continue, particularly where heatings are rare.

7.2.4 Treatment

Before effective treatment can be started, the site of the heating must be determined as accurately as possible. This requires a close examination of the mine plans, ventilation circuits, possible leakage paths and the daily record of CO gas concentrations at different locations. Once the site of the heating has been identified, the area must be sealed to cut off the supply of air to the heating. Mine layout and design should take into account the need to have suitable sites in solid, stable areas in which to construct seals. This may involve dividing the mine into discrete mining panels, each with a life less than the incubation period (if it is known) and such that each block can be isolated from the rest of the mine by seals. The sealing off method results in losses of reserves because of the need to leave pillars.

When seals are to be built in roadways through which methane drainage pipes pass, it is most important to have a contingency plan for dealing with these pipes. It is probable that drainage should continue as long as possible, disconnecting the pipes at the last possible moment. A section of the pipes can be left in place and used later for sampling or as part of a pressure balancing system. Decisions concerning methane drainage pipe ranges are even more important where open fires are concerned because the risks are generally greater; conditions change very rapidly and decisions must be made quickly, allowing little time for forethought.

Leakage of air past seals is a major problem. Apart from (or in addition to) grouting of the surrounding strata and sealants at the seal sites, pressure balancing can be used to equalize the pressure on the face of all the seals by ensuring open connections between them, or by pressure balancing chambers constructed at the seals. The pressure in the chambers is adjusted to equal the pressure behind the seal. This is done by either a small adjustable auxiliary fan or an adjustable venturi (The Institution of Mining Engineers, 1962).

A full and complete discussion on spontaneous heating, its causes, detection and control is a complex subject, too extensive to be adequately covered in this report. In addition, the circumstances in almost every case are different. It is more important to understand the basic principles and to use the experience gained by others.

The operation and ventilation of a mine liable to spontaneous combustion is one of the most difficult technical problems facing the mining engineer. Experience with heatings which have occured, and the documentation of events before and after the occurrence (particularly ventilation statistics), is vital for a proper understanding and prevention of future heatings.

7.3 OPEN FIRES

7.3.1 Causes

The causes of open fire have been well covered in many other papers and reports. The potential sources of ignition were classified by Thorp (1974) as:

a) the conversion of mechanical energy into heat; e.g. frictional heat generated by rotating picks striking rock or by defective conveyor idlers,
b) the conversion of electrical energy into heat; e.g. sustained arcing,
c) heat generated by chemical reaction; e.g. explosives, thermite reaction, smoking or spontaneous heating.

7.3.2 Effects

The relationships between ventilation and open fire are;

a) Ventilation dilutes and removes highly flammable methane from the mine.
b) Ventilation affects the progress of a fire by feeding oxygen to it and, in most cases, determines in which direction the fire will travel i.e. the fire will usually travel in the direction of air movement.

In addition, the ventilation is a major factor in determining whether a fire can be fought or must be sealed off.

c) Ventilation is affected by the fire because the asphixiating and toxic products of combustion are so rapidly carried into other parts of the mine. Smoke and gasses are responsible for the greatest loss of life in mine fires. Another important aspect which must always be remembered, especially in mines operating from steep slopes and in steep workings, is the chimney effect which can be created. When the hot air, smoke and fumes build up in a steep roadway, the roadway may act as a flue generating sufficient pressure difference to reverse the ventilation of the mine or create local ventilation reversals depending on the layout and circumstances of the mine in question.

7.3.3 Methane Ignitions

Methane is usually ignited by frictional sparking of cutting machine picks. There has been experience of this on the Cape Breton longwalls and in development headings in Alberta. The longwall ignitions take place in the poorly ventilated zone behind the machine at the cutting drum picks. This particular aspect is discussed in Sections 8.10 and 8.11 (d) of this report.

Ignitions in development headings are also mentioned in Section 9.5, "Potential for methane ignitions". The problem of dealing with methane without creating an airborne dust hazard is complex and is covered in the recommendations. However, ignitions in headings can at least be contained. Experience and full-scale testing in Poland and by The Safety in Mines Research Establishment in Britain have shown that rock dust barriers will effectively do this.

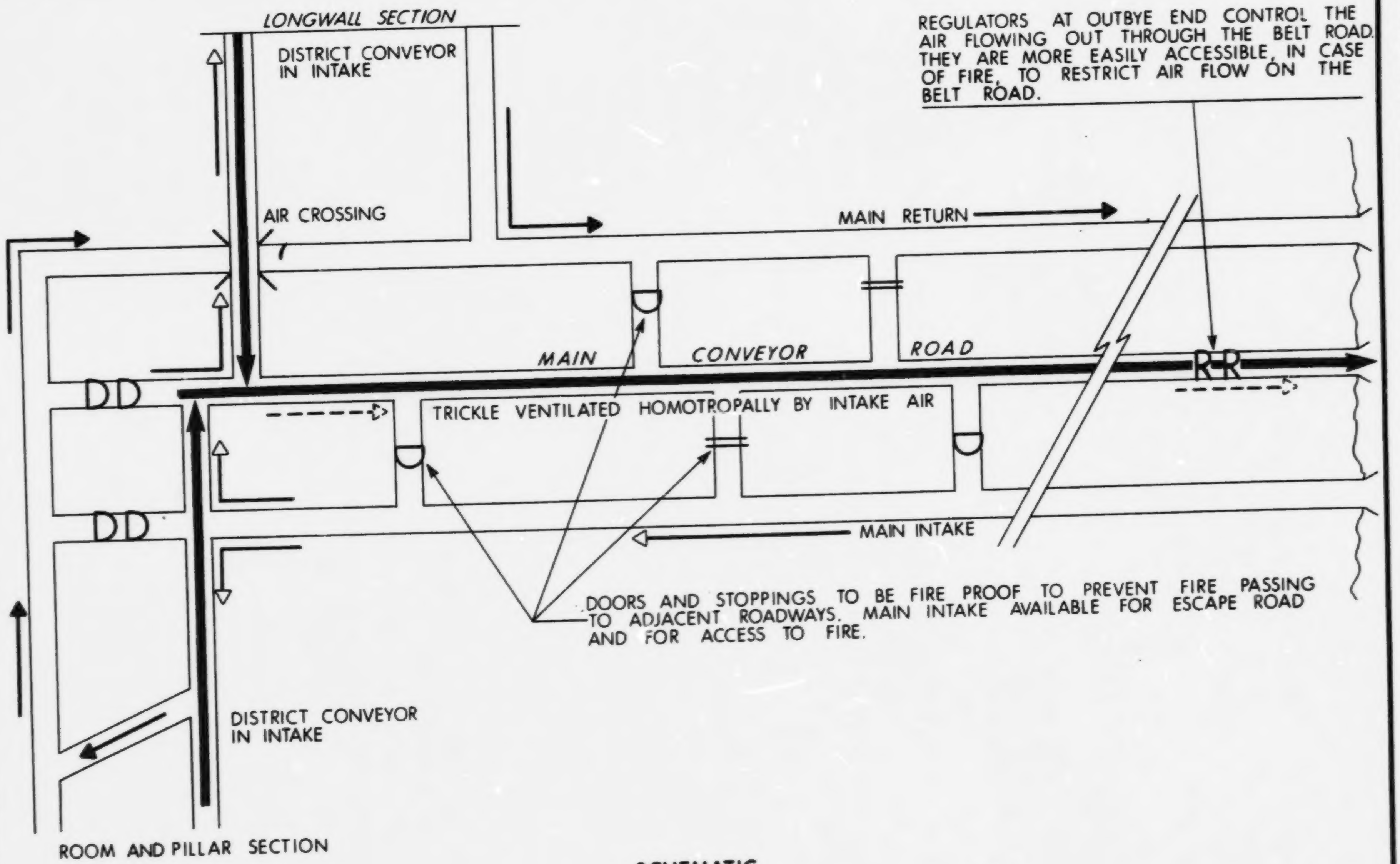
7.3.4 Main Belt Conveyor Roads

Main belt lines are not always frequently travelled during operational shifts, yet in most mines they are a source of fine coal dust. Mechanical and belt friction on belt conveyor systems are two of the main causes of fires in mines. Both give rise to frictional heat which often ignites coal dust in close proximity to the conveyor structure.

It is common practice for conveyors to be installed on the main intake airways. This means that, not only is the system antitropal, with airborne dust carried towards the working faces, but fire caused by a conveyor will tend to travel towards the face and render part of the roadway unuseable as an escape route. In addition, the smoke and fumes are carried to the working faces and the return airways, cutting off the remaining escape roads.

There are two arguments for the practice of placing main belts on intakes. The first is that, if the conveyor is a fire hazard, it should not be in return air carrying methane from the mine. This argument is debatable unless the mine is subject to outbursts or heavy methane emission rates. The second argument is that the face electrical switchgear is often on the conveyor road so that it too would then be in the return. Alternatively, to put the trunk conveyor in the return and the district conveyor in the intake would mean passing coal through doors, creating another dust problem as well as leakage.

It is suggested that these objections can be overcome by adaptations of the layout shown in Figure 7.1.


LONGWALL SECTION
DISTRICT CONVEYOR IN INTAKE
AIR CROSSING
REGULATORS AT OUTBYE END CONTROL THE AIR FLOWING OUT THROUGH THE BELT ROAD. THEY ARE MORE EASILY ACCESSIBLE, IN CASE OF FIRE, TO RESTRICT AIR FLOW ON THE BELT ROAD.
MAIN RETURN
MAIN
CONVEYOR
ROAD
R R
D D
D D
D
D
D
TRICKLE VENTILATED HOMOTROPALLY BY INTAKE AIR
MAIN INTAKE
DOORS AND STOPPINGS TO BE FIRE PROOF TO PREVENT FIRE PASSING TO ADJACENT ROADWAYS. MAIN INTAKE AVAILABLE FOR ESCAPE ROAD AND FOR ACCESS TO FIRE.
DISTRICT CONVEYOR IN INTAKE
ROOM AND PILLAR SECTION

SCHEMATIC
VENTILATION OF MAIN CONVEYOR ROADS

FIGURE 7.1

PART II

CANADIAN VENTILATION PRACTICE

INTRODUCTION

Data on Canadian ventilation practice was obtained mainly from visits to the operating mines and from discussions with technical personnel and mine management. This was supplemented by discussions with persons having knowledge and experience of the various mining areas and the ventilation of specific mines. The areas visited were in the Provinces of Nova Scotia, Alberta and British Columbia.

8. NOVA SCOTIA

8.1 MAIN UNDERGROUND COAL MINE AREAS

There are four areas in Nova Scotia in which underground coal mines are operating or are being planned at the time of writing. These are the districts of Springhill, Pictou, and the St. Rose district of Inverness county and the large mines located in the Sydney area of Cape Breton Island. There are three major active mines near Sydney, all owned and operated by the Cape Breton Development Corporation (C.B.D.C.). The No. 26 Colliery and Lingan Colliery mine undersea reserves using longwall advancing faces in the Harbour Seam. The Prince Mine uses short longwall retreat faces in the undersea reserves of the Hub Seam.

8.2 SPRINGHILL AREA (Includes information from N.S. Mines Inspector R. Henwood)

Coal has been mined underground in the Springhill area for many years going back to the 1870's. In more recent times, the three most important underground operations were the No. 2 Mine, the Syndicate Mine and the River Hebert Mine. The No. 2 Mine was entered by declines 4,360 m long, dipping about 20°. This mine was believed to be the deepest in North America at one time, operating at a depth of 1,324 m. Under these conditions, there were problems of gas and "bumps". A small proportion of the output was won from room and pillar mining and the rest from three hand-filled, retreating longwall faces 122 m long. Due to methane emissions, there was no blasting, all coal being mined by air picks. The mine was finally closed in 1958 following a disastrous "bump" in which seventy-five miners lost their lives.

The Syndicate Mine was opened in 1960 and finished about 1971-72. This mine was entered by slopes and mined a 1.5 m seam. Hand mining was used on both a longwall and a room and pillar section. The rooms were ventilated by line brattice. Total production was about 1000 tonnes per day.

The River Hebert Mine was the last operating mine in the Springhill area being closed toward the end of 1979. This was a small mine, entered by slopes, and had one longwall advancing face producing about 150 tonnes per day from a 1.0 m seam. It was said that this mine encountered some methane. Ventilation was from a single forcing fan delivering about 13 m^3/s.

At the moment there are no operating underground mines in the Springhill area though NOVACO has plans for a small underground operation, probably within the next two years.

8.3 PICTOU AREA (Includes Information from N.S. Mines Inspector R. Henwood)

Underground coal mining operations were carried out in the Pictou area from about 1828, notably the Foord Mine, sunk around 1866. This mine was known to have been troubled by methane but this could have been the result of poor ventilation practices. The mine was closed in 1892 (Dunn, 1982) The more recent underground coal mines were in the Stellarton, Thorburn and Westville districts.

At Stellarton, coal was mined from seams pitching at 20^o to 25^o and was exploited from slopes and from the Allen shaft. Operations ceased in the 1950's. At Thorburn, the McBean Mine closed in June 1972. This mine had produced from a single, hand-mined, longwall face in a 1.5 m seam. The mine was entered by slopes. Ventilation was produced by two fans sited at the top of one slope. The fans operated in parallel, exhausting 25 to 27 m^3/s. Spontaneous heating had occurred in this mine, probably due to poor mining methods and low recovery.

The only operating mine in the Pictou area is the Drummond mine at Westville. This mine is entered by slopes at about 20^0 incline and approximately 500 m long. The cover is only about 150 m. Coal is won by room and pillar methods using cutting and blasting. Between 30 and 40 men are employed in total. The mine produces only a small daily tonnage. Little or no methane is encountered and ventilation is produced by a single fan handling 12 m^3/s at a pressure of 187 Pa.

8.4 INVERNESS COUNTY

In Inverness county, on Cape Breton Island, the St. Rose mine started up in the early 1950's mining the No. 5 Seam. Between 30 and 35 men are employed underground to produce about 37,000 tonnes annually. The main fan is at a surface entrance to the mine and produces approximately 10.5 m^3/s at a pressure of about 310 Pa. The immediate ventilation problem is caused by the difficulty in maintaining the two access slopes, which serve as airways, through a mined out area. The two slopes are very confined and almost 600 m long. Stoppings and doors are constructed of wood and brattice but losses due to leakage are significant and only about 65% of the intake air reaches the bottom of the main intake slope. The proportion reaching the faces is much less. Small auxiliary fans and ducts are used to direct air into the development heading faces. Methane has never been a problem in the St. Rose area. If production levels were to be increased it is probable that dust would determine ventilation requirements rather than methane emission. The No. 5 Seam has not shown a tendency to spontaneous heatings.

Other small coal bearing areas on the west coast of Cape Breton have been mined in the past but there are apparently, no plans to reopen these areas.

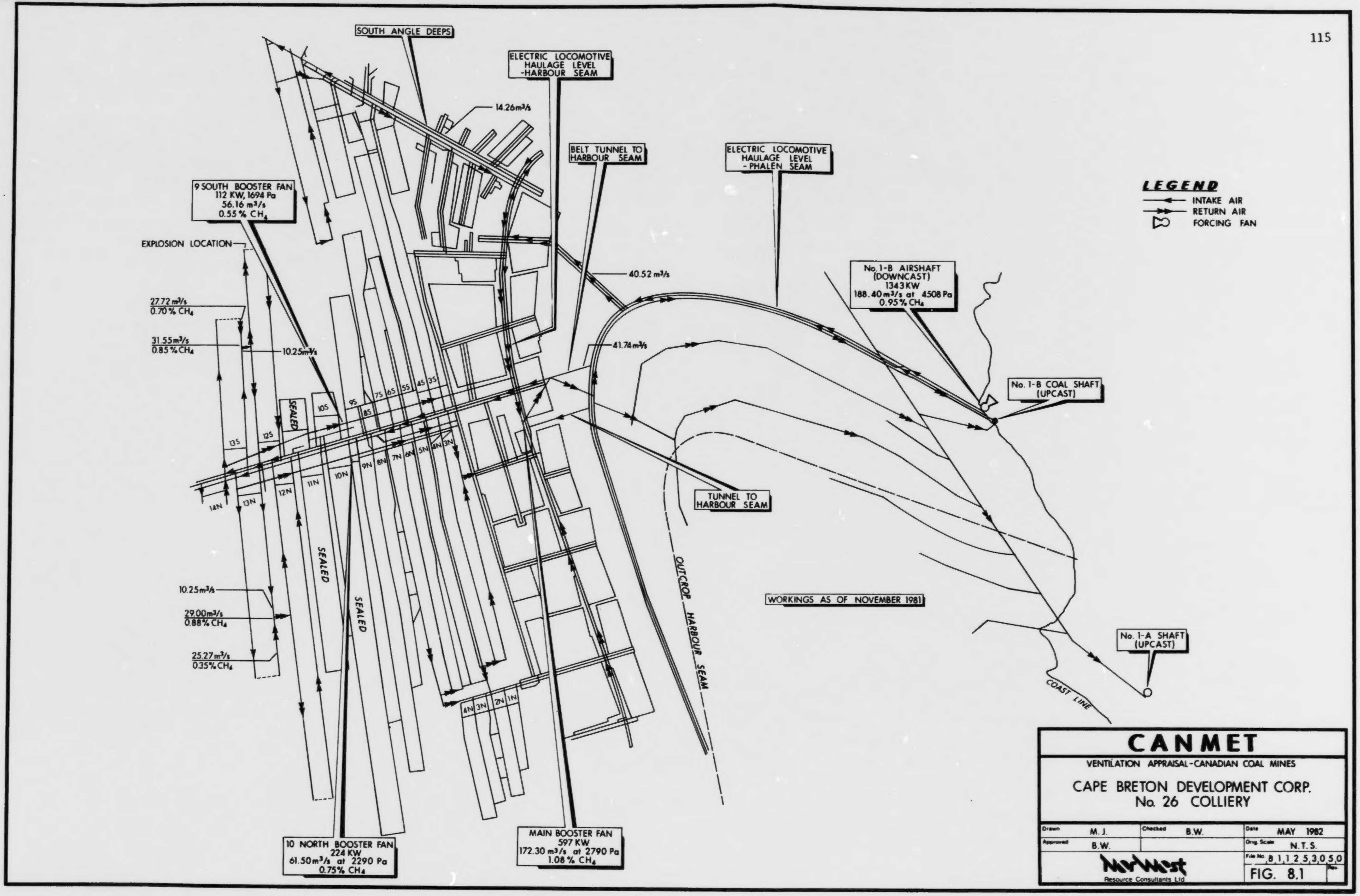
8.5 NO. 26 COLLIERY

This mine produces approximately 700,000 tonnes per year from two longwall faces and the associated development in the Harbour Seam. Extracted seam thickness is 1.8 m to 2.0 m and seam gradient ranges from 7.5 to 10 degrees. The mine is extensively developed and the furthermost face is approximately 8.8 km from the intake shaft at a depth of 750 m below the surface of the sea.

The mine is ventilated through one intake shaft by a Sirocco radial flow forcing fan handling 188 m^3/s at a pressure of 4.51 kPa. Three radial flow exhausting booster fans are also in operation. The main booster fan, sited 4.8 km from the intake shaft bottom, handles 172 m^3/s, at 2.79 kPa. The booster fans sited in 9 South and 10 North return airways handle 56 m^3/s at 1.69 kPa, and 61 m^3/s at 2.29 kPa, respectively.

Air is returned to the surface through two shafts which are located approximately 2.7 km apart. The total lengths of the respective ventilation circuits are 17.6 km and 20.0 km. Access to the Harbour Seam is gained via cross-measure roadways driven from the Phalen Seam at a location 4.27 km from the intake shaft bottom (Figure 8.1). Outbye of this point the old workings provide multi-path intake and return ventilation facilities. Inbye of this point the number of intake roadways varies up to three, and the number of return roadways varies up to four. For approximately 2 km down dip in the Harbour Seam, three intake roadways are flanked on each side by two returns.

At present, 13 North and 13 South longwall advancing faces, each approximately 200 m in length, are in production. These faces are ascensionally ventilated using the cundy system to control gob leakage, and the roadways of the adjacent face as returns. Airflows of approximately 26 m^3/s are present in the returns outbye of the brushing. Development headings for 14 North and 14 South faces are ventilated by 762 mm diameter auxiliary axial flow forcing fans with capacities of up to 7 m^3/s.


SOUTH ANGLE DEEPS
ELECTRIC LOCOMOTIVE HAULAGE LEVEL -HARBOUR SEAM
14.26 m^3/s
BELT TUNNEL TO HARBOUR SEAM
ELECTRIC LOCOMOTIVE HAULAGE LEVEL - PHALEN SEAM
LEGEND
INTAKE AIR
RETURN AIR
FORCING FAN
9 SOUTH BOOSTER FAN 112 KW, 1694 Pa 56.16 m^3/s 0.55 % CH$_4$
EXPLOSION LOCATION
40.52 m^3/s
No. 1-B AIRSHAFT (DOWNCAST) 1343 KW 188.40 m^3/s at 4508 Pa 0.95 % CH$_4$
27.72 m^3/s 0.70 % CH$_4$
31.55 m^3/s 0.85 % CH$_4$
10.25 m^3/s
41.74 m^3/s
No. 1-B COAL SHAFT (UPCAST)
SEALED
10S
9S
8S
7S
6S
5S
4S
3S
13S
12S
14N
13N
12N
11N
10N
9N
8N
7N
6N
5N
4N
3N
TUNNEL TO HARBOUR SEAM
OUTCROP HARBOUR SEAM
SEALED
SEALED
10.25 m^3/s
29.00 m^3/s 0.88 % CH$_4$
25.27 m^3/s 0.35 % CH$_4$
WORKINGS AS OF NOVEMBER 1981
No. 1-A SHAFT (UPCAST)
COAST LINE
4N
3N
2N
1N
10 NORTH BOOSTER FAN 224 KW 61.50 m^3/s at 2290 Pa 0.75 % CH$_4$
MAIN BOOSTER FAN 597 KW 172.30 m^3/s at 2790 Pa 1.08 % CH$_4$
NorWest Resource Consultants Ltd

CANMET

VENTILATION APPRAISAL-CANADIAN COAL MINES

CAPE BRETON DEVELOPMENT CORP.
No. 26 COLLIERY

Drawn	M. J.	Checked	B.W.	Date	MAY 1982
Approved	B.W.			Orig. Scale	N.T.S.
				File No.	8,1,1,2,5,3,0,5,0
				FIG. 8.1	Rev.

In addition to dilution by general ventilation, methane control is achieved through the application of cross-measure borehole drainage techniques. Both roof and pavement holes are used to capture approximately 0.5 m^3/s of methane from the two active faces. Methane concentrations in the ventilating current range from 0.3 to 0.7 percent in the return roadway outbye the brushing, to 0.6 to 0.9 percent in the cundy. Methane concentrations through the main booster fan and in the return shaft are approximately 1.1 and 0.95 percent respectively.

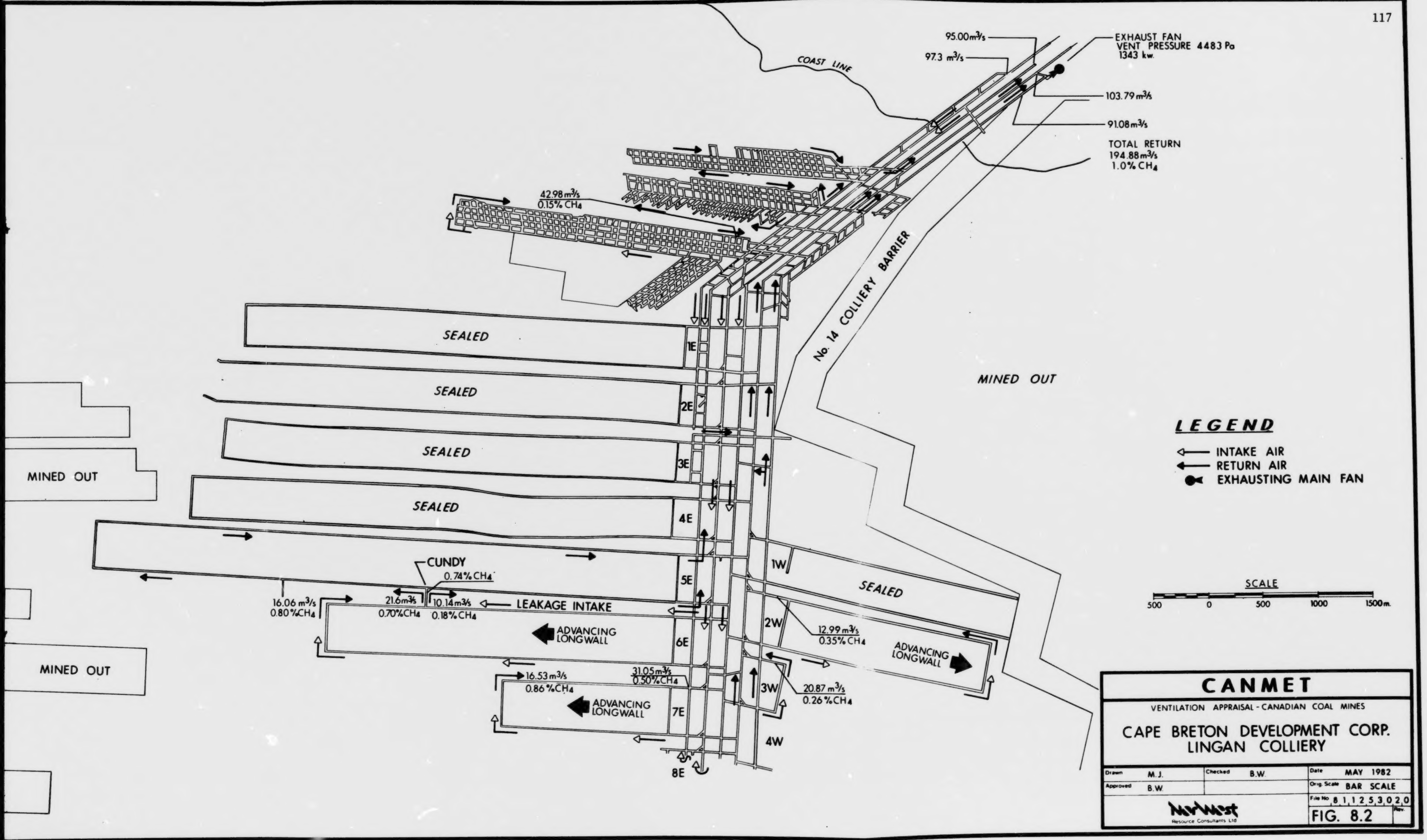
Total methane emission from the mine is 1.8 m^3/s which yields an emission rate of approximately 55 m^3/tonne of coal mined. The figure of 55 m^3/tonne mined includes emissions from older mines connected with the No. 26 Colliery ventilation system. An adjusted figure might be 35 m^3 to 40 m^3/tonne mined if only the methane from No. 26 Colliery itself was considered.

8.6 LINGAN MINE

This mine produces approximately 1,500,000 tonnes per year from three longwall faces and the associated developments, and one room and pillar section. All workings are in the Harbour Seam where the extracted seam thickness is 2.0 m to 2.2 m and the gradient averages 7.5 degrees. Lingan was developed in 1970 from three surface slopes which have since been augmented by a fourth to provide double intake and return roadways. The furthermost working face is 5.3 km from the surface and the cover below the surface of the sea is approximately 500 m.

The mine is ventilated by a radial flow exhausting fan handling 195 m^3/s at a pressure of 4.5 kPa. Each slope handles a similar volume of ventilation to a point approximately 1675 m from the surface, where a split of 43 m^3/s is used to ventilate the room and pillar section (Figure 8.2). Beyond this point as many as six dip roadways are used to provide ventilation. The longest ventilation circuit is approximately 10.7 km.

At present the three longwall advancing faces in production are 6 East, 7 East, and 3 West, each of which is approximately 210 m in length. The faces


95.00 m³/s
97.3 m³/s
EXHAUST FAN
VENT PRESSURE 4483 Pa
1343 kw.
103.79 m³/s
91.08 m³/s
TOTAL RETURN
194.88 m³/s
1.0% CH4
COAST LINE
42.98 m³/s
0.15% CH4
No. 14 COLLIERY BARRIER
MINED OUT
SEALED
1E
SEALED
2E
SEALED
3E
SEALED
4E
5E
MINED OUT
CUNDY
0.74% CH4
16.06 m³/s
0.80% CH4
21.6 m³/s
0.70% CH4
10.14 m³/s
0.18% CH4
LEAKAGE INTAKE
ADVANCING LONGWALL
6E
16.53 m³/s
0.86% CH4
31.05 m³/s
0.50% CH4
ADVANCING LONGWALL
7E
8E
MINED OUT
1W
SEALED
2W
12.99 m³/s
0.35% CH4
ADVANCING LONGWALL
3W
20.87 m³/s
0.26% CH4
4W
LEGEND
INTAKE AIR
RETURN AIR
EXHAUSTING MAIN FAN
SCALE
500
0
500
1000
1500 m.

CANMET

VENTILATION APPRAISAL - CANADIAN COAL MINES

CAPE BRETON DEVELOPMENT CORP.
LINGAN COLLIERY

Drawn	M. J.	Checked	B.W.	Date	MAY 1982
Approved	B.W.			Orig. Scale	BAR SCALE
				File No	8 1 1 2 5 3 0 2 0

NorWest
Resource Consultants Ltd

FIG. 8.2 Rev.

are ascensionally ventilated, with gob leakage controlled on 6 East by use of the cundy system which connects with the adjacent mined-out 5 East face. Airflows of 16 - 18 m^3/s are present in the return roadways outbye of the brushing. In the case of 7 East, which does not have cundy ventilation and has advanced 760 m, the airflow is 31 m^3/s at the outbye end of the return. The gob leakage flow accounts for approximately 45 percent of the total air available for this panel.

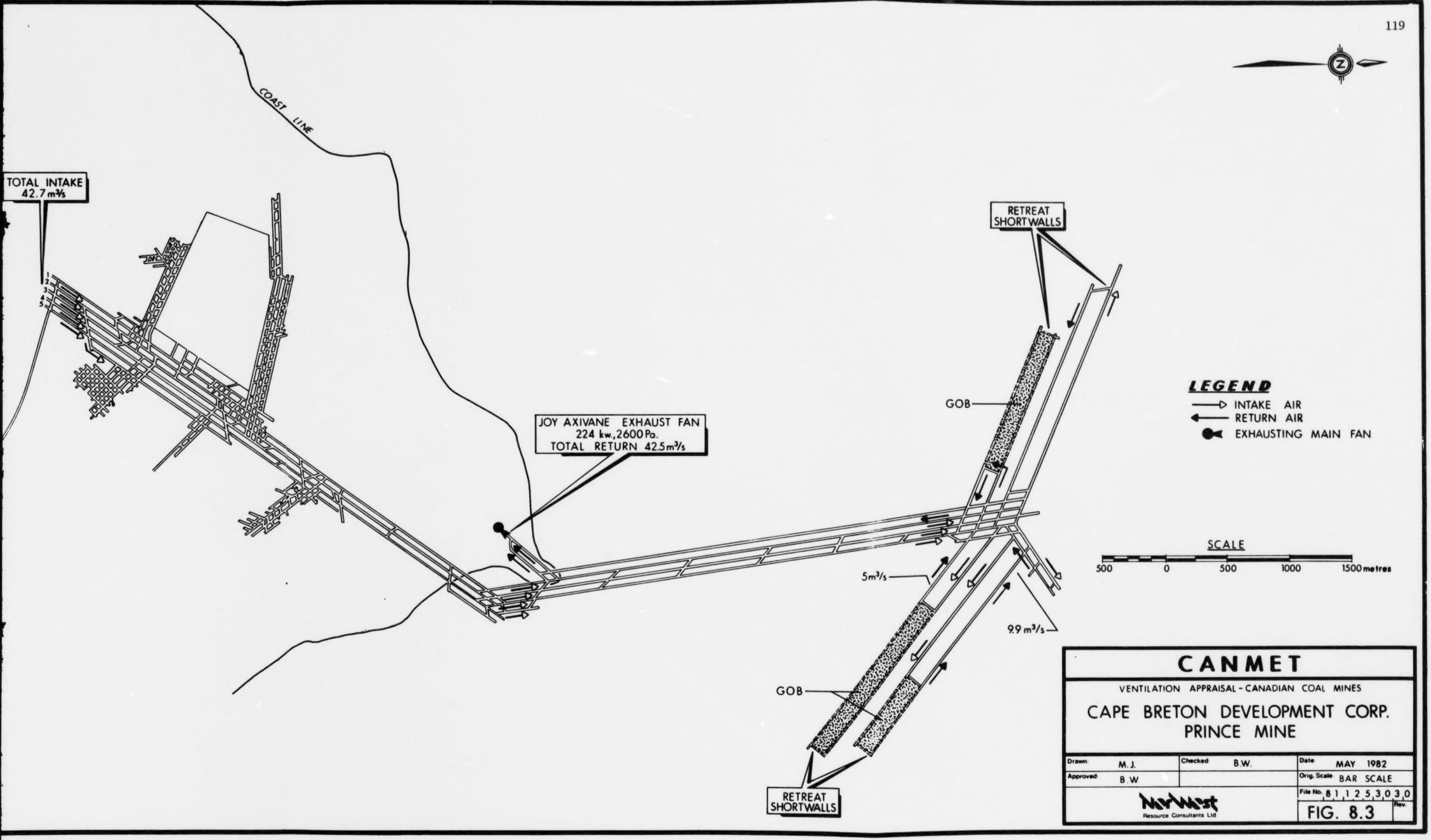
Methane control is achieved through dilution by general ventilation and drainage through cross-measure boreholes. Drainage results in the capture of approximately 40 percent of the total methane emitted. This capture rate has been significantly increased in the last three or four years. Methane concentrations in the general ventilation range from 0.8 to 0.9 percent in the return roadway outbye the brushing, to 0.6 to 1.0 percent in the slope returns.

Total methane emission for the mine is approximately 1.7 m^3/s which corresponds to an emission rate of approximately 23.5 m^3/tonne of coal mined.

8.7 PRINCE MINE

This mine produces approximately 500,000 tonnes per year from one short longwall retreating face and the associated developments in the Hub Seam which is 2.1 m thick. Seam gradient ranges from 5 to 7 degrees. During early mine life, room and pillar sections were developed to a limited extent (Figure 8.3). Current mining operations are located more than 5 km from the intake slope portal and three panels have been mined out.

The mine is ventilated by an axial flow exhausting fan handling 47 m^3/s at 2.6 kPa. As the return shaft is located near the coastline, approximately 2.1 km from the intake slope portals, the longest ventilation circuit is approximately 8.9 km. At least three, and as many as six roadways provide intake ventilation services while for 1.7 km a single return roadway is available.


COAST LINE
TOTAL INTAKE
42.7 m³/s
1
2
3
4
5
JOY AXIVANE EXHAUST FAN
224 kw., 2600 Pa.
TOTAL RETURN 42.5 m³/s
RETREAT SHORTWALLS
GOB
GOB
5m³/s
9.9 m³/s
RETREAT SHORTWALLS
LEGEND
INTAKE AIR
RETURN AIR
EXHAUSTING MAIN FAN
SCALE
500
0
500
1000
1500 metres
Z

CANMET

VENTILATION APPRAISAL - CANADIAN COAL MINES

CAPE BRETON DEVELOPMENT CORP.
PRINCE MINE

Drawn:	M. J.	Checked:	B.W.	Date:	MAY 1982
Approved:	B.W.			Orig. Scale:	BAR SCALE
	NorWest Resource Consultants Ltd			File No.	8 1 1 2 5 3 0 3 0

FIG. 8.3

At present the 2 East retreating face, which is 67 m in length, is nearing the end of its life. The face is descensionally ventilated with 10 m^3/s. Development roadways for 3 East retreating face are being advanced to a distance of approximately 1800 m prior to drivage of the face connection. Auxiliary ventilation is provided by an overlap system using an outbye forcing fan of 0.9 m diameter and an inbye exhausting fan of 0.6 m diameter.

Adequate methane control is obtained by dilution by the general ventilation. The Hub Seam is not particularly gassy and methane concentrations range from 0.7 percent in the face return roadways to 0.3 percent in the return shaft. Total methane emission for the mine is 0.14 m^3/s which yields an emission rate of approximately 6 m^3/tonne of coal mined.

8.8 VENTILATION SURVEYING AND MONITORING

Ventilation surveying procedures practiced in the underground mines of Cape Breton Development Corporation use traditional methods. Anemometry and direct area measurement are used to determine air volume flow rates, and periodic pressure surveys use gauge and tube methods. Methane concentrations are periodically measured at strategic locations in compliance with statutory requirements. The results of ventilation surveys are recorded, and analyzed for ventilation planning by digital computer facilities at the College of Cape Breton.

In addition to traditional techniques of monitoring the quantity and quality of mine ventilation, remote environmental monitoring is used at No. 26 Colliery and will be introduced at other operations of the Corporation. The existing scheme has been described by Schwartz (1981) and comprises monitoring of longwall face ventilation parameters, main and booster fan operating conditions, and drained methane particulars.

Air velocity and methane concentration are monitored at the outbye end of the longwall face intake and at the inbye end of the return. In order to maintain control over the cundy ventilation system, air velocities are also monitored at the outbye and inbye ends of the leakage intake. Main and booster fan inlet air velocities and methane concentrations are monitored in addition to bearing temperatures and vibration. Booster fan drive locations, which are separately ventilated by fresh air, contain a methane concentration transducer. Methane concentration and drained gas velocity are monitored on the inlet side of the methane drainage exhauster pumps.

Methane concentration transducers are Seiger BM1 pellistor monitors in the general ventilation, and Seiger BM 2H thermal conductivity instruments in the methane drainage pipe ranges. J-Tec air flow sensors, which measure the rate of vortex formation in the wake of a vortex generating rod, are used to measure air velocity. Fan pressure is monitored with a solid-state differential pressure transmitter.

Each transducer signal is fed to a remote station sited near the fan or strategically placed within the longwall panel. The remote stations, manufactured by Transmitton Ltd., can accept up to eight signals which are transmitted on a time division multiplex system to the surface master station. The surface master station contains the primary computer, an INTEL 8085 microprocessor, a video display console, keyboard and high-speed printer. Signals received by the primary computer may be accumulated and displayed in statistical or graphical form to indicate variations in each of the observed parameters. Audible and visual alarms are activated if a pre-determined value of any parameter is exceeded. Information can be formatted and recorded on magnetic or paper tape storage units, in addition to providing hard copy from the high speed printer.

8.9 METHANE CONTROL

Intensified application of techniques to control methane concentrations in the general ventilation is now evident at the Corporation's mines. Improved control has been obtained through increased ventilation efficiency, and enhanced methane capture by the drainage system.

The cundy system of ventilation has appreciably reduced gob leakage so that larger air quantities and higher velocities are available at the coal face. However, gob leakage still presents a problem inbye of the cundy due to the low resistance leakage paths through the gob. Roadside support comprises wooden cribs, augmented with hand-built stone packs where material is available, and these provide limited ventilation resistance. Leakage is substantially decreased by placing ureaformaldehyde, an organic foam known under the trade name of Isochaum, behind brattice cloth and between cribs on the high side of the intake roadway. Isochaum has little inherent strength and care has to be taken to maintain contact with the roof or the foam tends to slump.

Longwall face ventilation is enhanced by the substitution of 2-leg shield-type supports for chocks. Improved strata control has reduced bed separation and roof fractures, thus reducing methane emission from the roof and permitting improved capture by cross-measure drainage holes drilled near the face. In addition, face ventilation resistance is reduced because the shields present less obstruction than chocks, reduce convergence, and minimize gob flushing into the face area. Immediate roof collapse behind the shields reduces near-face gob leakage. The use of in-line brushing eliminates obstructions which are inherent with conventional brushing procedures, although the practice of dropping the brushing onto the face conveyor reduces the availability of packing material.

Outbye of the face, ventilation efficiency is improved by prompt salvaging of abandoned places and effectively sealing these off.

Methane drainage is achieved through cross-measure boreholes drilled in the return roadway to lengths of approximately 40 m. Roof holes are drilled at 20 m intervals inclined at 55 degrees to the horizontal. Pavement holes are drilled in a direction parallel to the line of the roadway. Standpipe lengths average 9 m and are sealed with densotape and grout. Holes are maintained within 20 m of the face to improve capture efficiency. Approximately 15 to 18 holes are on suction simultaneously with the gas feeding into a 200 mm diameter pipe range. Depending on requirements, up to three Nash-Hytor HL6 pumps, sited at the outbye end of the return or leakage intake, are maintained at suction pressures of up to 40 kPa. Suction pressure at each borehole is controlled to ensure maximum flow at an acceptable purity. Capture rate averages 40 percent, and 60 percent has been attained. Drained methane is released into the general ventilation and diluted outbye in the main return.

8.10 CUNDY VENTILATION

Although the use of the cundy system of ventilation was not new to the Corporation's operations, more intensive use of the technique was introduced following the Report of the Commission of Inquiry into the explosion at No. 26 Colliery in 1979 (Elfstrom, 1980).

Normal operating practice allows approximately 30 m^3/s and 10 m^3/s of air to enter the main and leakage roadways respectively. Outbye of the cundy, gob leakage is reduced to a negligible level. Inbye of the cundy, the amount of leakage depends on the distance between it and the faceline. Air reaching the intake face end splits to provide direct face ventilation and near face leakage behind the powered supports. The use of shield-type supports, and consistent face advance, promotes regular roof collapse with the minimum of near-face leakage. Generally, 20 m^3/s of air is available to dilute methane emission at the power loader; this flow rate is approximately double that available before the introduction of cundy ventilation. The benefits of increased airflow and air velocity are considerable in maintaining methane concentrations within acceptable limits, and in minimizing methane accumulations and layers in hazardous locations.

In general terms, a methane emission balance for a longwall panel ventilated in the conventional manner may be compared with a cundy ventilated face, by reference to air volume flow rates and methane concentrations. Total methane emission comprises that present in the general ventilation and that in the drained gas. Whilst conventional ventilation was in use, approximately 1.8 percent of methane was present in 30 m^3/s at the outbye end of the face return. The methane flow rate in the general ventilation was, therefore, 0.54 m^3/s, approximately one-half of which issued from the gob. The drained gas volume was approximately 0.4 m^3/s, at a methane purity averaging 50 percent, yielding a methane volume rate of 0.2 m^3/s. Total methane emission from the longwall face was, therefore, 0.74 m^3/s. When cundy ventilation is in use, the cundy handles 40 m^3/s containing 1 percent of methane which yields a methane flow rate of 0.4 m^3/s. The drained gas contains 55 percent methane in a total flow of 0.6 m^3/s or 0.33 m^3/s of methane. Total methane emission from the cundy ventilated face is, therefore, 0.73 m^3/s.

It is apparent that the total methane emission from a longwall advancing face and asssociated gob is almost constant irrespective of the method by which the face is ventilated. However, the sources of emission vary. The methane volume which migrates from the gob into the return of the conventionally ventilated face is present, in the case of the cundy ventilated face, as approximately equal volumes of methane in the general ventilation and in the drainage system. The former is adequately diluted to safe limits by the additional airflow which is available, and the latter results in an improved methane drainage capture rate. In the circumstances considered, methane drainage capture rate has been increased from 27 percent to 45 percent. It may be concluded from the above analysis that, although the cundy system promotes conditions conducive to enchanced methane drainage capture, the corollary is equally true in that the success of the cundy system is dependent on the application of an effective methane drainage system.

8.11 MINE FIRES

Precautions against mine fires, fire-fighting practices, and experiences with mine fires at the Corporation's mines were the subject of a previous study (Dames & Moore and H.G. Stephenson, 1979). Two major incidents have occurred in recent years. The fire which occurred on No. 11 South Wall, No. 26 Colliery, on June 19, 1975, was described by LeBlanc and Phalen (1976).and Ea
Details of the explosion which occurred on No. 12 South Wall, No. 26
on February 24, 1979, are contained in the report of the Commission of Inquiry (Elfstrom, 1980). In both instances the majority of the evidence indicated that methane was ignited by incendive sparks created by the picks of the power loader cutting into quartzitic sandstone.

A geological peculiarity affecting the Harbour Seam is the presence of sandstone intrusions whose nature, direction, extent and frequency are unpredictable. They may occur in the bottom, middle or top of the seam, affecting a local area of the face or extending over 30 m (Eaton, 1976). Although the quartz content of the intrusions vary, it is commonly high enough to classify the rock as having an intermediate or high incendive temperature potential. Unless the zones containing the intrusions can be prepared by blasting or other means, it is inevitable that, at some time, they will be contacted by the picks of power loaders. It is, therefore, necessary to consider both the minimization of potentially incendive sparks and the presence of methane in the vicinity of any sparks which occur.

Research on the minimization of sparks through reduced peripheral pick speeds and the application of water is documented by Powell, et al. (1975). High face airflows and velocities, although necessary to avoid methane layering and the propagation of flame from the ignition point at the pick across the face area into the gob, do not, per se, result in adequate methane dilution at the location where the coal is cut. Methane desorption tests on the Harbour Seam indicate low permeability and slow release of gas, consequently a large proportion of the methane produced by the Harbour Seam is released at the face, with the amount increasing as coal size decreases during cutting. Additional observations indicate that, where the normal shale roof of the seam is replaced by sandstone intrusion, the impermeable nature of this material provides a cap and prevents the normal release of methane ahead of the face during stressing and destressing in the front abutment zone. The likelihood of producing an ignition of gas by an incendive spark is therefore increased due to increased methane emission.

It is difficult to provide sufficient fresh air to adequately dilute methane emitted at the point of cutting. This is particularly true in the case of the leading drum of a power loader which is cutting in the same direction as the ventilation, because the area is shrouded by the machine, the cut, and solid coal. The use of hollow shaft venturis, together with the flow of air promoted by water issuing from pick-face flushing, may effectively provide adequate dilution. However, partly functional venturi ventilators may result in an inadequate airflow introducing only sufficient oxygen to transform an extinctive atmosphere into one containing a large volume of explosive gas. The desirability of using venturi ventilators is, therefore, a moot point and is contingent upon fully efficient operation at the designed air volume flow rate.

9. ALBERTA

9.1 MAIN UNDERGROUND COAL MINE AREAS

In Alberta there are two areas in which coal is, or has been, mined by underground methods: the Mountains/Foothills region, on the east slopes of the Rockies, and the Plains region extending eastwards from the Foothills.

In the Mountain-Foothills region, geologically disturbed strata has resulted in severe faulting, folding and steep seams, with depths of cover ranging from zero to over 600 m. The coal is generally of a bituminous quality with isolated zones of semianthracite.

In the Plains region, the seams are flat-lying and overlain by flat topography (hence there is little variation in cover). The quality of the coal ranges from subbituminous to the lower end of the bituminous scale. The Plains coals are of a lower rank than those in the Mountains/Foothills region and are more prone to spontaneous heating.

Methane emission rates in Mountain and Foothill areas range from moderate to extremely high. In Plains areas, methane emission rates are generally very low.

Coal seams in the Mountains and Foothills are generally friable and surrounded by rocks which range from weak to strong. On the Plains, coal seams are hard and prominently cleated; the roof and floor strata are generally weak to moderate in strength.

In the Plains areas the main centres for underground mining in recent years have been the Star Key mine area near Edmonton, the Drumheller and East Coulee areas, and the Lethbridge area. At the moment, only the Atlas Mine at East Coulee and the more recent pilot mine at Kipp near Lethbridge are ventilated and open for inspection. The main centres in the Mountains/Foothills areas are Crowsnest Pass, Canmore, Nordegg, Luscar/Coalspur, and Grande Cache. At this time, underground mining is only active at the Cardinal River Coals' mine at Luscar (which is under development) and the underground operations of McIntyre Mines near Grande Cache.

9.2 THE ATLAS MINE (Plains)

This mine was exploiting a 4 m thick, flat seam by a room and pillar system. Access to the coal from the surface is by a rectangular vertical shaft about 135 m deep. The shaft was used for coal hoisting and as an upcast return. Air is forced into the mine by a small, axial flow fan mounted at the top of an inclined rock tunnel. The fan is capable of producing 28 m^3/s at 0.87 kPa though it was not required to handle more than 17 m^3/s at any time during the life of the operation. In winter the air is heated by a burner converted from coal to natural gas. Although the coal is low rank there have been no underground spontaneous heatings, probably because the remaining coal pillars are very competent and do not crush.

The Atlas mine was opened in 1956 at a time when a large number of operations were producing coal in the Drumheller area. The mine was placed on standby in 1979, and no firm plan exist to recommence production at this time. The Atlas mine is typical of the Drumheller region though the seam is appreciably thicker than in most of the adjacent operations.

9.3 THE KIPP MINE (Plains)

The Kipp Mine is located in southern Alberta west of Lethbridge. Access to the coal is by a 4.25 m diameter vertical shaft, sunk in 1980. The Galt Seam is approximately 2 m thick and lies at a depth of 204.0 m. A total of 202 m of development headings have been driven as pilot test drivages. The headings are ventilated by means of a 1.2 m diameter raise shaft to the surface. At the top of this shaft is an axial flow Engart forcing fan with an air heater. Headings were ventilated by a 3.3 m^3/s exhausting auxiliary fan. A small amount of methane was found, particularly near faults. Concentrations of up to 0.25% were measured in the faces of the headings.

Conditions in the Kipp Mine are typical of those encountered in several operations in the Lethbridge area. Although methane emissions were generally low, at least one ignition leading to loss of life is known to have occured at the Coalhurst Mine. The consultants are not aware of any problems with spontaneous combustion underground in the Lethbridge area though many of the refuse piles and outcrops show evidence of heatings.

9.4 LUSCAR (Foothills)

Cardinal River Coals Ltd. are developing 11½° rock tunnel slopes down to the thick, steep, Jewel Seam. This mine is being planned to extract the coal by hydraulic methods at a depth of cover down to about 400 m. There have been no previous underground workings at this depth in this area. 'CANMET' services were enlisted to determine the methane content and rate of desorption from the coal at this depth (Canmet Report ERP/MRL 80-116 (TR)). An estimate of anticipated methane quantities has been made based on coal permeability and will be used for ventilation planning. The accuracy of the predictions will only be determined by actual mining experience.

As a result of the CANMET investigations, in which desorbtion tests were carried out on coal samples from two boreholes, the methane content of the coal was found to be at the 256 m depth, 6,600 dm^3/tonne and at the 408 m depth, 17,700 dm^3/tonne. After four hours, the rate of methane desorbtion was in the region of 10 dm^3/tonne/min. but desorbtion rates considerably higher than this might be expected when the coal is freshly cut at the faces. How much higher is not known, neither is the effect of strata pressures resulting from depth and tectonic forces. Historically, in the Rocky Mountain Coal Mining Areas, very high methane emissions have been experienced, particularly at depths in excess of about 500 m when, under certain conditions, outbursts occurred in which large volumes of methane were released. This kind of phenomena still cannot be predicted or controlled. In Western Canada very large amounts of coal lie at these depths. Effective ventilation will be important for successful underground mining of these reserves. The experience at Cardinal River illustrated the need to continue the research into assessing the factors affecting methane desorbtion rates and the application of test results to ventilation planning, particularly in virgin areas.

9.5 GRANDE CACHE (Mountains)

At the time the consultants visited this operation there were five active underground coal mines, all owned and operated by McIntyre Mines Ltd. Total production from this group of mines is approximately 1,035,000 raw metric tons of metallurgical coal annually. Room and pillar mining is used, employing continuous miners, shuttle cars and conveyor belt haulage.

Of the potentially mineable seams present in the series, three have been worked to date: No. 4 Seam (6 to 8 m thick), No. 10 Seam (3 to 4 m thick), and No. 11 Seam (1.5 m to 3 m thick). Seam inclinations reach 80 degrees, but to date only those reserves inclined at less than 18 degrees have been mined to any extent. Table 9.1 shows details of the mines worked during 1980.

TABLE 9.1

MCINTYRE MINES
SEAMS WORKED DURING 1980

Mine	Seam	Thickness (Metres)	Annual Tonnage - 1980 (Metric-Raw)	Remarks
9A	4	6-8	109,000	Development
9G	4	6-8	Negligible	Commenced late 1980
2-11	11	1.4-3.4	239,000	Development & Depillaring
2R-11	11	1.4-3.4	77,000	Development
2R-4	4	6-8	525,000	Depillaring
5-4	4	6-8	115,000	Development & Depillaring

Depth of workings from surface reaches 600 metres.

9.5.1 Main Fans

All mines employ an exhausting system of main ventilation with some using large diameter (1.37 m) auxiliary fans in clusters and others using a single, large axial flow, main fan. Typically, each of the Engart fans within a cluster passes 28 m^3/s at a pressure of 600 Pa. A 2.44 m diameter axial flow fan handles up to 123 m^3/s at a pressure of 2000 Pa.

9.5.2 Intake Air Heaters

During colder weather, it is necessary to heat the main intake air. Heaters are fueled either by natural gas or propane. Naked flames from a series of gas jets impinge against a baffle plate located in the path of the main intake air stream. Because McIntyre employs exhausting main fans, the larger heating units are placed in chambers offset from the main intake portals so as not to impede the passage of personnel and materials. A major advantage of locating the heating units in this way is that all of the heat generated enters the mine, carried by the intake ventilation current.

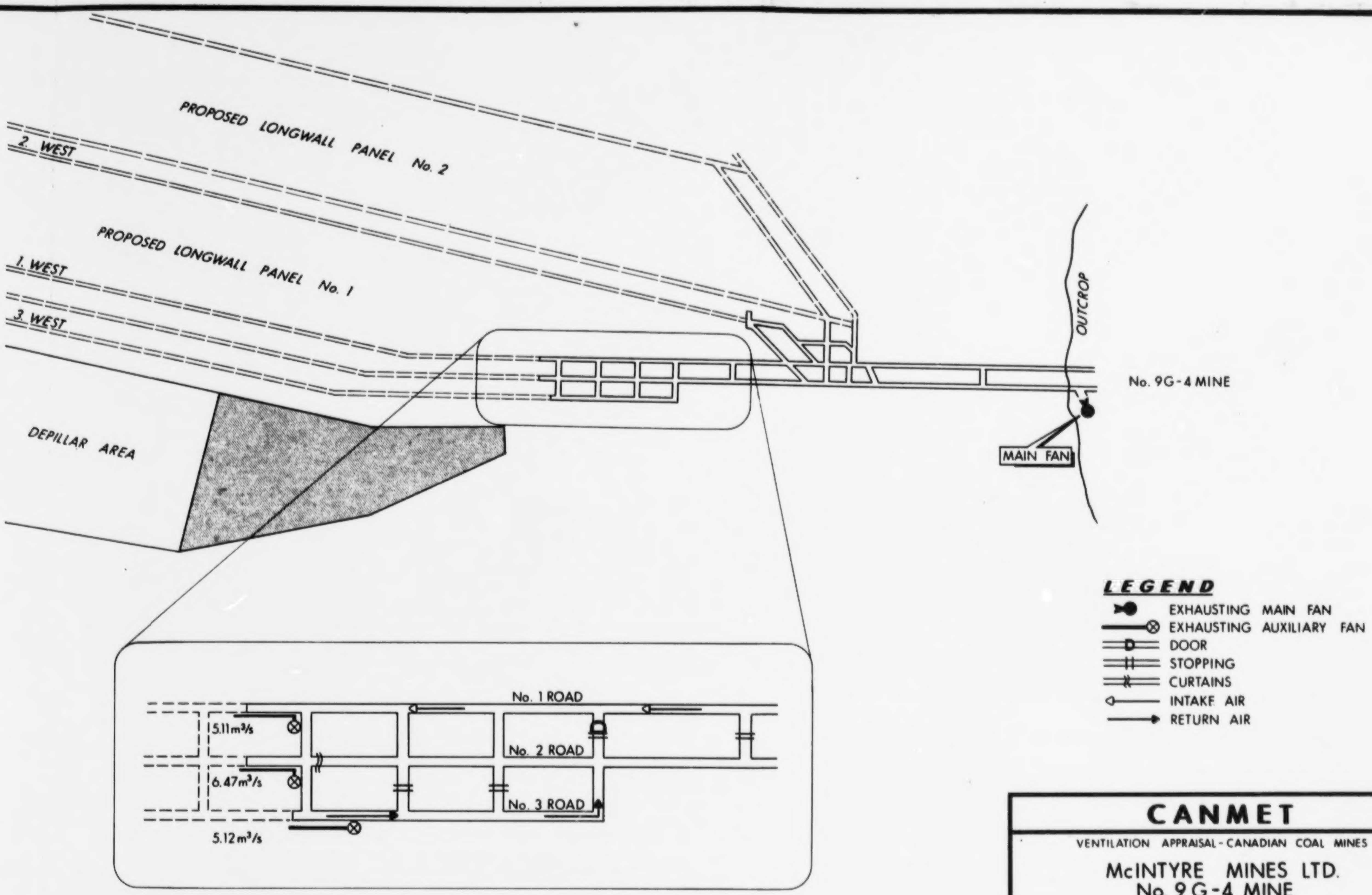
9.5.3 Mine Layouts (See Figure 9.1)

Mine layouts typically comprise a number of main entries extending into the reserves from outcrops. Submains are struck off in appropriate directions from the mains to divide the take into mining sections within which production panels are formed. Some, such as Mine No. 5-4, are extensively developed, while others, such as Mine No. 9G-4, are in early stages of development. (See Figure 9.1) At all mines, main intake airways parallel main returns. There are no vertical shafts or booster fans at the McIntyre operations.

9.5.4 General Methane Control and Ventilation Survey Techniques

Methane control is by dilution in the main air body. In gassy development headings, auxiliary forcing fans and ducting are used; in less gassy headings, an exhausting auxiliary system ventilates the face. Methane emission rates exceeding 1.5 m^3/min. have been found in some production headings at Mine No. 5, Seam No. 4. Air quantities at individual sections vary from 1.3 m^3/s to 5.2 m^3/s. Ventilation monitoring techniques include routine sampling and measuring of the air at strategic points within the ventilation system. MSA Spotter and MSA D6 hand-held methanometers are used for periodic measurements, while continuous measurement is provided by methane monitors, fitted with visual read-out and electrical interlock, on continuous miners. A tube bundle analysis and monitoring system is being introduced. Initially, this system will sample CO and CH_4 at strategic locations.

Ventilation surveys have been carried out at McIntyre using the tube and inclined gauge method.


PROPOSED LONGWALL PANEL No. 2
2. WEST
PROPOSED LONGWALL PANEL No. 1
1. WEST
3. WEST
DEPILLAR AREA
OUTCROP
No. 9G-4 MINE
MAIN FAN
No. 1 ROAD
No. 2 ROAD
No. 3 ROAD
5.11m³/s
6.47m³/s
5.12m³/s
LEGEND
EXHAUSTING MAIN FAN
EXHAUSTING AUXILIARY FAN
DOOR
STOPPING
CURTAINS
INTAKE AIR
RETURN AIR

CANMET

VENTILATION APPRAISAL - CANADIAN COAL MINES

McINTYRE MINES LTD.
No. 9 G-4 MINE

Drawn	M.J.	Checked	B.W.	Date	MAY 1982
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NorWest Resource Consultants Ltd.				File No.	8 1 1 2 5 3 0 4 0
				FIG. 9.1	Rev.

9.5.5 Potential For Methane Ignitions

McIntyre's systems of testing, monitoring and recording of methane emissions are, in general, adequate to detect methane during normal operations. However, certain areas of the No. 4 Seam are overlain by a cap rock that, when struck by continuous miner picks, has produced incendive sparking sufficient to ignite methane at the face of headings. The cap rock sometimes fell immediately after the coal was removed during the cutting cycle. If the continuous miner is used to break and load out this cap rock, it can result in frictional sparking under the cutter heads where concentrations of methane are not easily detected. The mode of auxiliary ventilation was changed from exhausting to forcing so that turbulence in the face would prevent methane accumulations and layering.

9.5.6 Dust

Since the change from exhausting to forcing auxiliary ventilation at No. 5 Mine, there has been some difficulty in maintaining dust standards within statutory limits. McIntyre has tried a Gundlach dust extractor system on two continuous miners and, though the device met with some success, operational difficulties resulted in eventual disuse.

The two Gundlach dust extractor/scrubbers were fitted as an option by Lee Norse, the air intake portion of the devices placed at the centre of each machine head. The suction fans for the extractors were hydraulically powered from a power take-off unit (PTO) on the continuous miners. The devices functioned such that the dust laden air from beneath the cutting heads was drawn through the Gundlach air intake and passed through a ring of water sprays. Subsequently the wetted dust was discharged as a black sludge onto the continuous miner conveyor chain and the cleaned air recirculated into the ventilation air stream.

Dust reduction was noticeable but not completely effective. The equipment proved to be reliable with little down time. The system appeared to operate better in a blind heading than in an open pillar, probably because the face and ribs of the heading confined the airborne dust close to the continuous miner extraction system. The apparatus was tried on and off for a period of two years (1979-80) with promising results.

However, there were problems inherent with the system:

- The operating crews objected to the high noise levels.
- The air intake were located under the cutting heads of the miners and frequent blockages occurred from cut coal (Lee Norse was asked to relocate the air intakes to the sides of the miner head but they showed little enthusiasm.

Overlap and controlled recirculation methods of auxiliary ventilation have been studied but not tried. Airborne dust has not been a serious problem in pillar workings.

9.5.7 No. 2 Mine Conveyor Tunnel

Raw coal from the open pit and some coal from the underground mines is crushed at a surface breaker station west of No. 2 Mine. The product is then transferred by a belt conveyor, located in a tunnel cut through No. 2 Mine, delivering to the cleaning plant. Keeping this tunnel free of fine coal dust has been difficult. Much of the dust originates at the coal breaking phase of transportation and it has been suggested that research be directed toward coal treatment at the breaker station as a means of dealing with the problem.

9.5.8 Stoppings and Doors

To date, stoppings have been built from cinder blocks and from plywood with the plastic sheet seal. In both methods, Uniseal is sprayed on exposed surfaces to complete the seal. Unfortunately the stoppings have to be built near the working faces in unstable ground conditions, one result being that the stoppings are damaged by strata movement and leakages occur.

Production officials at the mines indicated that they would be interested in improved stoppings that would take less time to erect, would be recoverable and would be able to accept moving ground conditions. There is interest in the Kennedy type of metal prefabricated stopping (See Section 6.1).

An hydraulically operated, prefabricated, "American Mine Door" air lock has been installed at the main fan portal of 9G-4 Mine.

9.5.9 Spontaneous Combustion

The McIntyre operations do not have a history of spontaneous combustion.

9.5.10 Future Ventilation Plans at McIntyre

McIntyre is currently studying the feasibility of reintroducing longwall mining. Ventilation aspects to be considered include whether the panels would utilize a system of bleeder returns, homotropal or antitropal direction of ventilation on the face, and whether single entry or multi entry methods of panel development will be used. The possibility of 2000 m long single entry development drivages on auxiliary ventilation is being considered.

An increased use of remote monitoring and control of ventilation is planned. To date, services from the U.K. National Coal Board have been used to carry out ventilation network evaluations.

9.6 OTHER UNDERGROUND OPERATIONS (ALBERTA)

Two underground operations in the Rocky Mountain area of Alberta closed two years ago. Although they are not likely to reopen, they are of interest because the difficult ventilation problems encountered may be experienced in future operations in Alberta or British Columbia. The Coleman and Canmore mines were similar in many respects. Both were originally operated by drilling and blasting from the solid and transporting coal by a combination of chutes, conveyors and mine cars. Both operations switched to continuous miners and shuttle cars in the 1960's. Canmore later abandoned continuous miners for a semimechanized system of slushers and chain conveyors.

In both mines, methane emission rates were high to extremely high (65 m^3/tonne at Canmore) and the extremely friable coal was very dusty. In some areas, particularly beyond a depth of cover of about 220 m, certain seams were liable to outbursts. In both mines, at certain times, ventilation of development drivages in coal was severely hampered by gas emissions and the inability to dilute methane to satisfactory levels by auxiliary ventilation of headings. In general it can be said that, while these difficult conditions are by no means typical of Rocky Mountain seams, they are certainly not uncommon.

Shortly before these operations closed. serious consideration was given to a trial of methane drainage by in-seam drilling or cross-measure boreholes. It was felt that application of some form of methane control, other than dilution using a general ventilation system, might be necessary in future operations.

10. BRITISH COLUMBIA

10.1 MAIN UNDERGROUND COAL MINE AREAS

There are three principal areas in British Columbia where underground coal mining is being, or has been, practiced:

a) Vancouver Island, in the vicinity of Nanaimo and Cumberland. The mines were once extensive but ceased operations over twenty-five years ago.

b) In northeastern British Columbia. An underground mining prospect in the Sukunka area, where exploratory headings have been driven in from the outcrop, is now indefinitely suspended.

c) In southeastern British Columbia in the Fernie-Sparwood area, close to the Alberta border. B.C. Coal is currently operating two underground coal mines in conjunction with a very large open pit at their Sparwood operations. Underground operations have, in the past, been very extensive in this area.

In addition to the areas mentioned above, small quantities of coal have been mined by underground methods in the Tulameen and Princeton coalfields. Mining operations in the Rocky Mountains and Foothills of British Columbia encounter a similar wide range of conditions to the mines in the mountainous areas of Alberta.

10.2 B.C. COAL - BALMER NORTH MINE

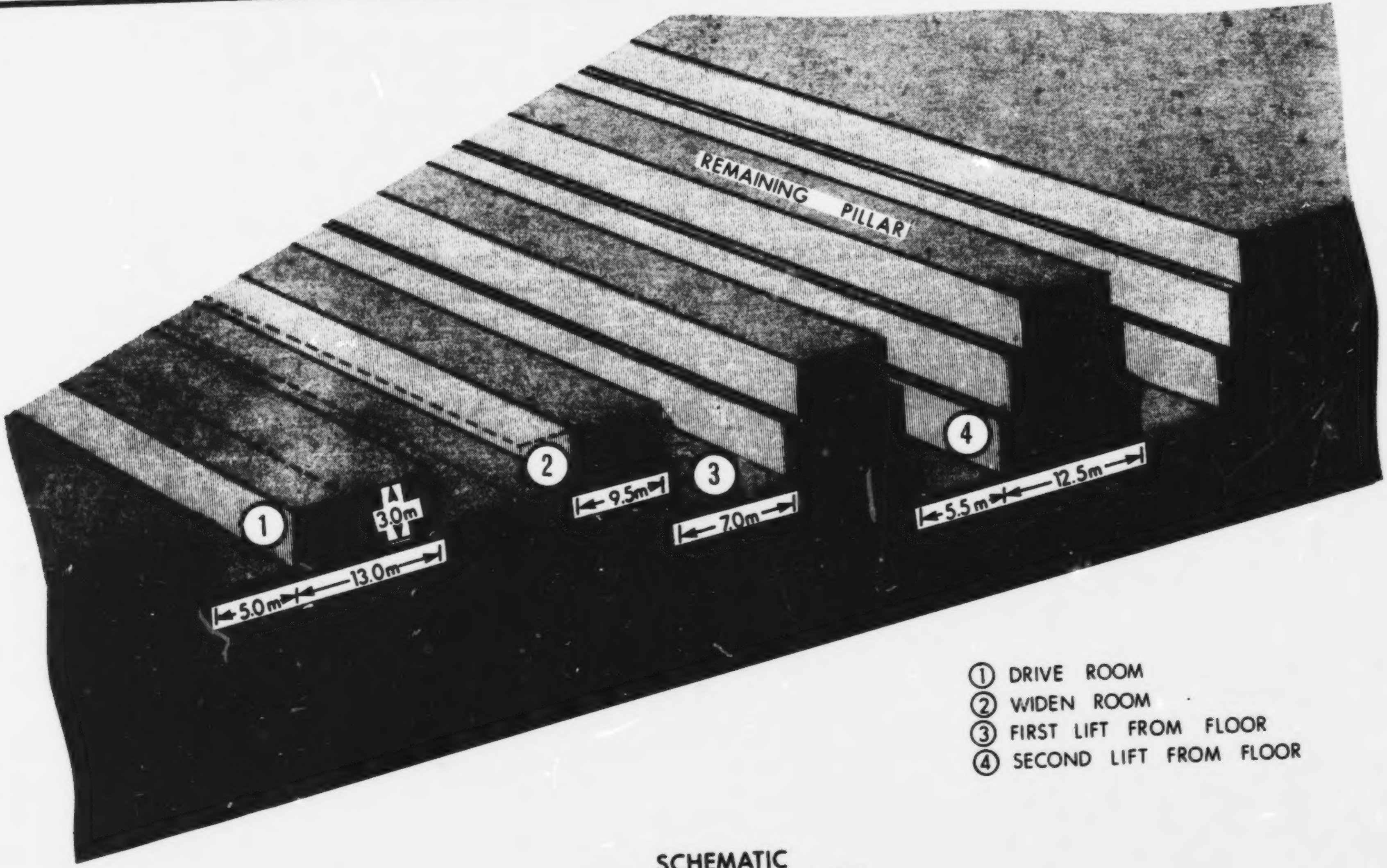
This mine is producing about 170,000 tonnes/yr with one continuous miner/shuttle car unit delivering onto a belt conveyor system. The coal being mined is the No. 10 Seam with a thickness of about 15 m. The mining system is partial extraction by room and pillar. Initially, rooms 5.0 wide and 3.0 m high are developed at the roof level of the seam and supported by roof bolts. The pillars so formed are about 25 m long by 12 or 13 m wide. In the extraction stage, these pillars are reduced to about 9 or 10 m in width by widening the original room. The coal is then taken from the floor of this wide area in two lifts, the first about 7.0 m wide and the second about 5.5

m wide. The remaining pillar is then about 12.5 m wide at its base. These pillars do not crush, therefore the gob areas remain open, though inaccessible due to floor heave and roof deterioration. The total height excavated in the rooms is about 10 m (See Figure 10.1).

Balmer North is ventilated by a 300 kW, axial flow, exhausting fan sited at the top of an unlined shaft. This fan exhausts 95 m^3 at a pressure of 1500 Pa. Main access to the seam is via two short rock tunnels, one for a coal conveyor and a second for supplies transportation. This second tunnel is fitted with a natural gas heater at the intake of a secondary forcing fan. The fan is used to comply with B.C. Safety codes which require a minimum positive air flow through the heater at all times. Air-lock doors are installed at the portal and the heater fan only assists by handling a part of the total mine intake. Additional unheated intake air is drawn ino the mine through the coal conveyor portal by the larger, exhausting, main fan.

The ventilation circuit in Balmer North is not complex. Conveyors are located on the intake roadways; ventilation of development headings and rooms is by auxiliary forcing fans and air ducting. The return is via roadways leading to the main ventilating fan shaft. Up to 1967, the pillar mining system practiced in Balmer North involved the excavation of long, narrow, high openings. Ventilation at the machine was by auxiliary fan and ducts. Little air flowed through the gob cavities and velocity was very low due to the cross-section of the cavity.

In 1967, a gob cave displaced a quantity of methane into the operating roads and the gas was ignited. There was an explosion in which fifteen miners lost their lives (B.C. Mines and Petroleum Resources Report, 1967, p. 420). As a result of this explosion the old Balmer North pillaring system was changed. Pillars are now retreat mined in shorter lengths. The mined-out sections are stopped off from time to time, directing a sufficient air flow through gob cavities adjacent to the coal getting operations. In this way, undiluted methane does not flow into the active working areas.


REMAINING PILLAR
1
2
3
4
3.0m
5.0m
13.0m
9.5m
7.0m
5.5m
12.5m
① DRIVE ROOM
② WIDEN ROOM
③ FIRST LIFT FROM FLOOR
④ SECOND LIFT FROM FLOOR

SCHEMATIC
BALMER NORTH MINE
SEQUENCE OF EXTRACTION

FIGURE 10.1

10.3 B.C. COAL - BALMER SOUTH HYDRAULIC MINE

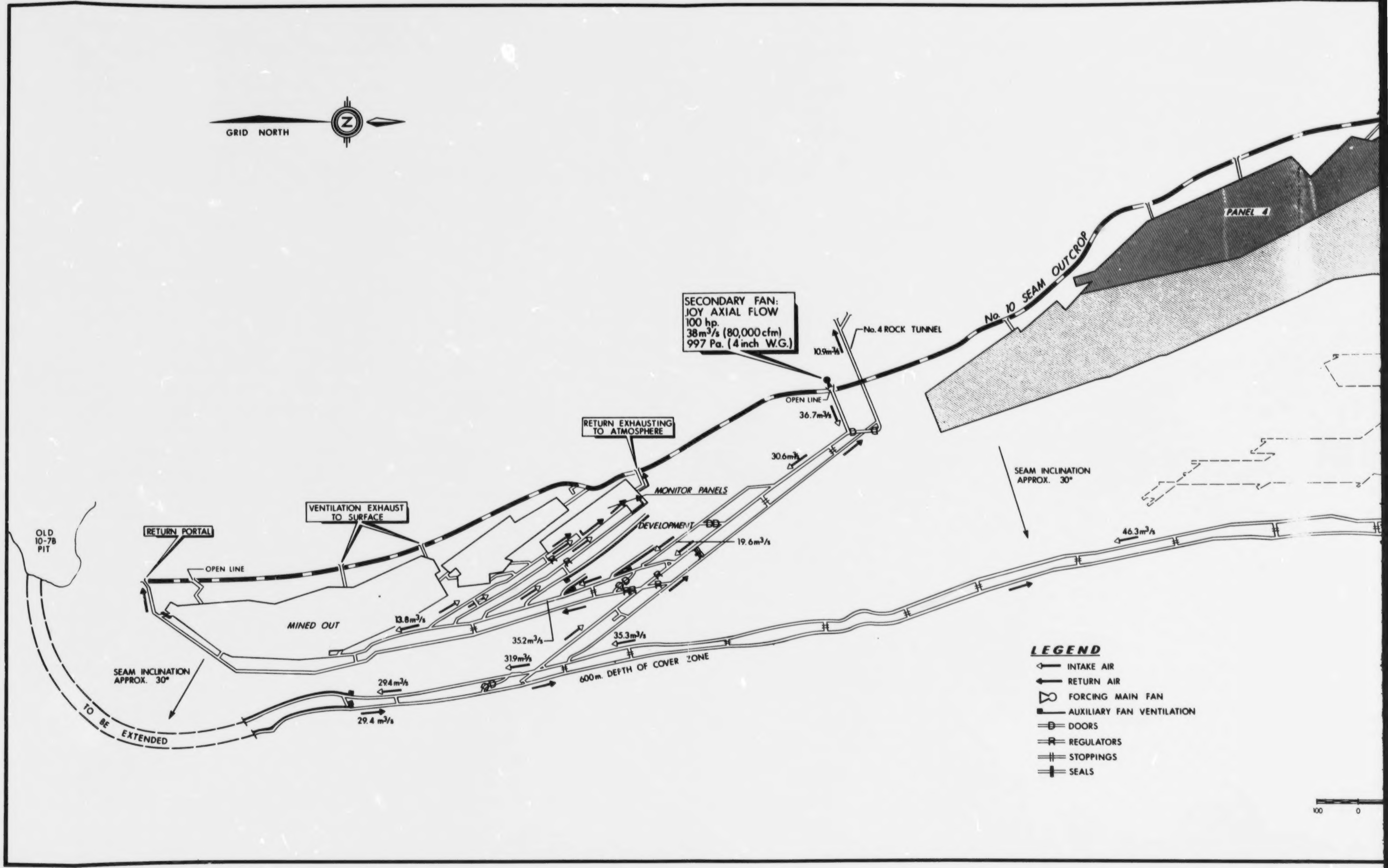
Several papers have been published describing this operation (Grimley 1974; Parkes and Grimley, 1975).

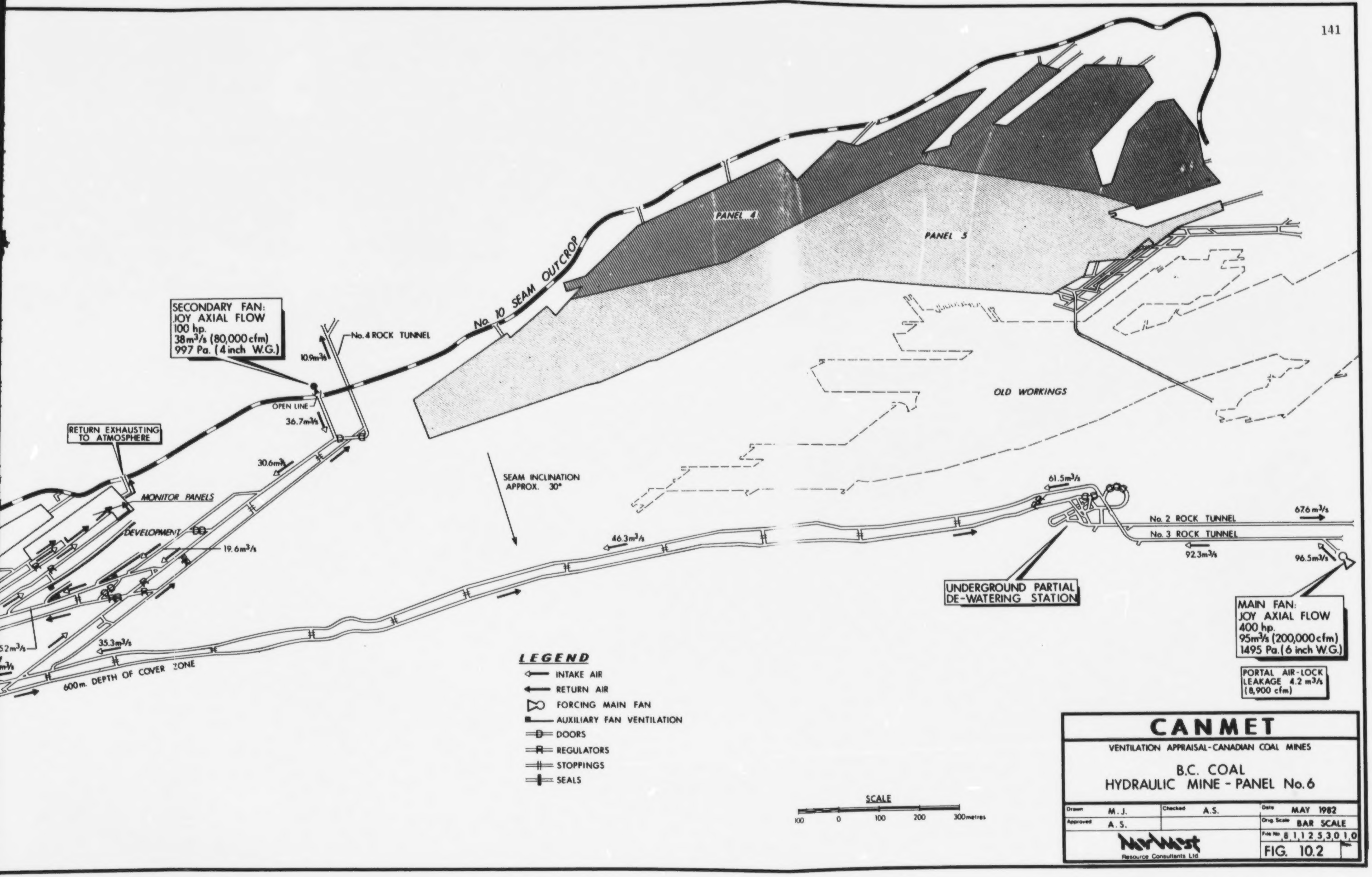
All hydraulic mining at B.C. Coal is in the No. 10 Seam in the Balmer South area. This type of mining was started experimentally in 1967-70 and proved successful. Part of the present Panel 6 Hydraulic Mine lies to the dip side of the first hydraulically mined areas in Panels 3, 4 and 5. Access to Panel 6 is by No. 2 and No. 3 rock tunnel declines, at the foot of which is an underground coal/water handling plant. Here, coal flumed from the working sections is partially dewatered by screening. The screen overflow is transported to the surface by belt conveyor and the slurry underflow is pumped up to a surface dewatering plant. Figure 10.2 shows the layout of the Panel No. 6 Mine.

10.3.1 Mining Conditions and General Layout

The No. 10 Seam averages 15 m in thickness and is inclined at an angle of about 30° from the horizontal. The area being mined is long and narrow, approximately 3,650 m long in a north-south direction, and 370 m wide. The access tunnels and underground dewatering plant are at a northeast corner. From the dewatering plant, two main entries rise at about 5° in a roughly southerly direction and, at the south end, they curve west towards the outcrop. The coal to be mined from panel 6 is at the rise side of the main entries. In the southern half of the mine area, the seam outcrops at the surface. Above the northern half of the mining area, the earlier hydraulic panels No. 4 and 5 cut off Panel 6 from access to the surface outcrop. Some old conventional room and pillar workings, where the upper part of No. 10 Seam has been partially mined out, also lie in this northern area. It is expected that monitor sublevels will be developed in the lower part of the seam and that the remaining coal can be mined hydraulically.

The main entries were developed in coal, commencing from the area of the dewatering plant. At this stage, one entry served as an intake and the other as a return; cross-cuts were made at approximately 240 m intervals and the


GRID NORTH
Z
SECONDARY FAN:
JOY AXIAL FLOW
100 hp.
38m³/s (80,000 cfm)
997 Pa. (4 inch W.G.)
No. 4 ROCK TUNNEL
10.9m³/s
No. 10 SEAM OUTCROP
PANEL 4
OPEN LINE
36.7m³/s
RETURN EXHAUSTING TO ATMOSPHERE
30.6m³/s
MONITOR PANELS
DEVELOPMENT
19.6m³/s
SEAM INCLINATION APPROX. 30°
46.3m³/s
VENTILATION EXHAUST TO SURFACE
RETURN PORTAL
OLD 10-7B PIT
OPEN LINE
MINED OUT
13.8m³/s
35.2m³/s
35.3m³/s
31.9m³/s
29.4m³/s
29.4 m³/s
600m. DEPTH OF COVER ZONE
SEAM INCLINATION APPROX. 30°
TO BE EXTENDED
LEGEND
INTAKE AIR
RETURN AIR
FORCING MAIN FAN
AUXILIARY FAN VENTILATION
DOORS
REGULATORS
STOPPINGS
SEALS
100
0


No. 10 SEAM OUTCROP
PANEL 4
PANEL 5
OLD WORKINGS
SECONDARY FAN:
JOY AXIAL FLOW
100 hp.
38m³/s (80,000 cfm)
997 Pa. (4 inch W.G.)
No. 4 ROCK TUNNEL
10.9m³/s
OPEN LINE
36.7m³/s
RETURN EXHAUSTING
TO ATMOSPHERE
30.6m³/s
MONITOR PANELS
DEVELOPMENT
19.6m³/s
SEAM INCLINATION
APPROX. 30°
46.3m³/s
35.3m³/s
52m³/s
m³/s
600m. DEPTH OF COVER ZONE
61.5m³/s
No. 2 ROCK TUNNEL
67.6 m³/s
No. 3 ROCK TUNNEL
92.3m³/s
96.5m³/s
UNDERGROUND PARTIAL
DE-WATERING STATION
MAIN FAN:
JOY AXIAL FLOW
400 hp.
95m³/s (200,000 cfm)
1495 Pa. (6 inch W.G.)
PORTAL AIR-LOCK
LEAKAGE 4.2 m³/s
(8,900 cfm)
LEGEND
INTAKE AIR
RETURN AIR
FORCING MAIN FAN
AUXILIARY FAN VENTILATION
DOORS
REGULATORS
STOPPINGS
SEALS
SCALE
100
0
100
200
300 metres

CANMET

VENTILATION APPRAISAL-CANADIAN COAL MINES

B.C. COAL
HYDRAULIC MINE - PANEL No. 6

Drawn M. J.
Checked A.S.
Date MAY 1982
Approved A. S.
Orig. Scale BAR SCALE
File No. 8 1 1 2 5 3 0 1 0
NorWest Resource Consultants Ltd

FIG. 10.2

faces of the advancing headings beyond the last cross-cut were ventilated by auxiliary fans and ducts. Continuous miners, shuttle cars and a temporary conveyor system were used for the drivages. Support was by fully covered steel arches. During the main entry development, there was difficulty with methane as the faces approached the 600 m cover zone. Methane emissions up to 48,000 m^3/day or 0.6 m^3/s, were encountered. Auxiliary fans (56 kW) and ducts were utilized to deliver 9.5 to 14 m^3/s of air to the face of the headings.

The large air quantities needed at the face to dilute the methane led to a problem of keeping airborne dust at acceptable levels. A number of possible solutions were examined including machine-mounted extractors, and spray patterns used to direct air flow in the face. A Mining Research and Development Establishment dust filter was installed but proved to be too big and cumbersome. No fully satisfactory solution was developed during the problem period.

Just beyond the limits of the old hydraulic panels, two more entrances to the seam were made; one is an outcrop opening where a 75 kW, axial flow, fan is mounted. This fan is currently forcing an air quantity of 38 m^3/s into the mine and developing a pressure of 1000 Pa. In association with this outlet, the No. 4 rock tunnel was driven to intercept the seam. From these outlets two entries were driven downhill at about 12^o to join with the main entries. As they approached the deep cover zone, they did not encounter the high gas emission rates experienced in the main entries.

The lower main entries are still being advanced to the outcrop in the 10-7B pit area. The present mine layout is shown on the plan (See Figure 10.2); three sublevels are equipped with monitors and three sublevels are under development. The plan also shows the ventilation circuit and the quantities of air flowing.

10.3.2 Main Fans and Heaters

Main Fan:	Axial flow, 300 kW
Located At:	No. 3 Rock tunnel - surface
Duty:	95 m^3/s, 1500 Pa

Heater:	8.8 MW output Natural gas open flame
Secondary Fan:	Axial flow, 75 kW
Located At:	No. 4 rock tunnel area, at surface of a short inclined shaft
Duty:	38 m^3/s, 1000 Pa

10.3.3 Instrumentation and Control

The main and secondary fans are fitted with vibration and temperature sensors at the main bearings. The heaters have CO detectors just down stream of the burners. Until recently, B.C. Coal was operating a Tube Bundle mine atmosphere monitoring system to determine CO content in air samples continually drawn from selected locations underground. The samples pass through an automatic analyser with a daily print out of the results. The apparatus was part of a CANMET project with the analyser housed in a mobile trailer. At present, the only location where the system could be usefully applied is of short duration and has no suitable power supply. The system is to be re-established in the near future at the No. 4 rock tunnel access area. An important function of the Tube Bundle system at the hydraulic mine is to provide early warning of the development of spontaneous heating by indicating CO gas percentages. The system proved capable of providing an early warning of the heating in Panel No. 5 in 1978.

B.C. Coal has also instituted a monitoring system which detects main fan stoppages. A warning signal is transmitted by radio to a central security office. Warnings are picked up by the repeated scanning of a radio monitoring system. It is possible that the scope of this equipment will be increased to cover additional aspects of the main ventilation system.

10.3.4 Ventilation of Development Headings

Development headings are ventilated by auxiliary fans forcing through 90 cm flat bag type air ducts. The fans used are 56 kW axial flow and 37 kW B70's with bifurcated cases which allow the motor to be located outside the air flow of the fan. The auxiliary system delivers from 9.5 to 14 m^3/s to the

a) it must be as leak-proof as possible
b) it must be constructed as quickly as possible (a large seal has taken as much as three man-months to complete)

Seals are cut well back into the sides and floor, after the steel arches have been removed and loose roof coal taken down, the seal may be 6 to 7 m wide and can be as much as 5 m high. From 2000 to 3000 bags of concrete are required to construct a seal 45 to 75 cm thick. Even so, leakage in the range of 1 - 2 m^3/s can be expected. The best seal was obtained in a dip roadway; the rise side of the seal was accidentally flooded with silty water. The water drained away leaving a silt seal about 8 m long which effectively cutoff all leakage. To speed up the construction of seals, B.C. Coal has just received delivery of a concrete placing machine.

10.3.9 Ventilation Monitoring and Computer Use

Routine ventilation monitoring is achieved using conventional anemometers, velometers, water gauges and magnahelic gauges; the latter are portable as well as a permanent feature at main fans. In addition there is a barometer and thermometer for recording the surface atmospheric conditions. The above instruments are used to meet the Provincial Mine Regulations.

To monitor underground air quality, the following instruments are used:

a) Tube Bundle sampler and ecolyzer for CO (the tube sampler is temporarily out of service - weekly manual sampling is being used)
b) Methanometers - hand-held, also machine-mounted, constant running
c) Flame Safety Lamps
d) Humidity - Thermometers and hygrometers
e) Multi-gas detectors - i.e. drager tubes and samplers (particularly used for routine diesel exhaust tests)
f) Airborne dust samplers - Bendix personal samplers
  - MRE 113A units, area samplers

For ventilation survey and investigative tasks the following additional instruments are maintained at the mine:

a) Pitot tube assembly
b) Inclined water gauge and portable magnahelic gauges and
c) Precision altimeters (an accuracy of ± 0.5 m).

B.C. Coal has the services of an in-house computer suitable for running ventilation network programs. It is hoped that programs can be developed to take into account changing conditions of airways and gobs. The equipment will also be useful for planning of new mine ventilation systems. The programs used by B.C. Coal were purchased from the Pennsylvania State University. One program was based on an airflow network and a second permitted the introduction of various levels of methane injections into the network. Attempts were made to establish a representative model for Panel No. 6 and Balmer North Mines by establishing mine roadway and gob friction factors from pressure and quantity surveys. The results proved satisfactory in the case of Balmer North Mine but unsatisfactory for the Panel No. 6 Hydraulic Mine. A N.C.B. ventilation engineer, (Mr. Barry Lord, 1981) had reviewed the Panel No. 6 ventilation system to determine whether ventilation changes could reduce the progress of oxidation at a heating to a safe level and enable mining operations to continue without loss of coal barriers or operating time. This of course is not easy to do when mining a 15 m thick seam and Mr. Lord made no recommendations for changes to the present Panel No. 6 ventilation system.

11. RECOMMENDATIONS FOR RESEARCH AND DEVELOPMENT

11.1 INTRODUCTION

The recommendations for research and development are grouped as follows:

a) those which pertain to Nova Scotia,
b) those which pertain to Alberta,
c) those which pertain to British Columbia, and
d) those which are related to Canadian underground coal mining in general.

11.2 NOVA SCOTIA

Considerable improvements have been achieved in the mine environment at the C.B.D.C. mines since the explosion at No. 26 Colliery on February 24, 1979. However, the implementation of new ventilation techniques and the introduction of remote monitoring have identified topics requiring further investigation.

a) Cundy Ventilation

The introduction of cundy ventilation has appreciably improved face ventilation by reducing gob leakage. With extended experience of this ventilation technique, the issues which now require to be investigated are:

i) The optimum spacing between cundies to minimize gob leakage inbye of the last cundy.
ii) Utilization of alternative roadside sealing methods (e.g. pump and aqua packing) to increase the resistance of the leakage path through the gob. In addition, the use of alternative methods for the disposal of brushing debris into the pack hole should be investigated.
iii) Detailed analysis of the methane emission balance in the cundy ventilated panel in order to more precisely identify the sources of methane emission and proportion of gas which issues from each.

b) Methane Sources

The introduction of remote continuous monitoring gives rise to an extensive data base from which to determine methane emission sources and patterns for the whole mine. Methane drainage techniques can then be evaluated for optimum methane capture.

c) Remote Monitoring

In order to derive maximum benefit from the system, the following issues should be addressed:

- i) Optimum location of air velocity and methane concentration transducers in relation to their geographical disposition within the mine, and their specific location at a point in the mine roadway.
- ii) Derivation of air volume flow rates from spot recording velocity transducers.
- iii) Method of presentation of recorded data to management to secure effective control with minimum effort and production of paperwork.

d) Ventilation of Cutting Zone

Effective and adequate general face ventilation, in view of the low permeability of the Harbour Seam, direct attention to methane emission and dilution in the specific, problem area of the cutting zone. Research is required into:

- i) Relative efficiency of methods of ventilating the cutting zone with regard to leading and trailing drums, and the direction of cutting relative to the direction of air flow.
- ii) The use of desorption tests to assist in assessing the methane content of the coal where intrusions are present to determine emission in these locations.
- iii) The pattern of dust emission from shearer drums.

e) Mining Layout

The cundy system of ventilation has inherent disadvantages previously identified in this report. Alternative mining layouts should be investigated to:

i) Determine the applicability of longwall retreat mining to the prevailing geologic and mining conditions and existing layout.

ii) Assess the impact on methane emission and ventilation requirements of pillar and panel longwall advancing layouts and to evaluate the effect of pillar width.

f) Methane Drainage

Improvements in cross-measure drainage techniques and equipment have significantly improved capture rates. Remaining areas of investigation include:

i) Enhancing methane capture from strata below the seam by experimentation with pavement holes of varying depth and direction.

ii) Potential for methane recovery from sealed areas by drainage from behind pressure controlled stoppings.

g) Offshore Service Shafts

The possibility of using artificial islands or offshore platforms to construct mine shafts closer to reserves located some distance from the coastline should be examined. Recent developments resulting from the development of offshore oil and gas reserves may provide a basis for adaptation of these techniques to coal development. Offshore shafts would be particularly beneficial in providing improved ventilation for reserves more than six km. from the shoreline. Another advantage, not related to the subject of this report, would be shorter man riding times helicopters were used to convey personnel to the shafts.

The recommendations listed above are aimed primarily at the present and future operations of the Cape Breton Development Corporation in the Sydney coalfield. Other underground coal operations in Nova Scotia, present and planned, do not appear to have serious ventilation problems that cannot be solved by an application of normal ventilating practice.

11.3 ALBERTA

Several of the following recommendations related to conditions in the Rocky Mountains of Alberta could be of equal interest, in similar conditions in British Columbia.

a) Ventilation in Long, Single Development, Headings

As mentioned previously, both forcing and exhausting methods of auxiliary ventilation are in use at McIntyre Mines. Forcing auxiliary fan ventilation improves climatic conditions, prevents methane accumulating in the face but can create an airborne dust problem. On the other hand, where exhausting ventilation is used, the dust problem is reduced but methane accumulation can occur. There are advantages to developing retreat longwall panels by single entry drivages if a suitable means of ventilation can be found. The use of overlap and controlled recirculation techniques to deal with the dust and methane problems and to improve climatic conditions should be investigated.

b) Dust in Mechanised Development Headings

The use of a Gundlach dust collection system has already been described (See Section 9.5.7). The results appear to justify further trials of this or similar equipment.

c) Use of Coal Dust Treatment

At McIntyre Mines , dry coal from the open pit, crushed at the surface breaker station, is contributing to the coal dust accumulations in the conveyor tunnel which cuts through No. 2 Mine. This type of problem is not limited to McIntyre's operation. Consideration should be given to research into methods of treating the coal at the breaker station before it enters the underground conveyor tunnel. McIntyre Mines is preparing to test a foam dust suppressant at its raw coal stockpile towers.

d) Particular Problems in Ventilating Thick Seams

In both Alberta and British Columbia this has been a problem. The control strata and ventilation of workings (and gobs) are very closely related in thick seam mining. One item in particular is the development of a system for determining the extent and pattern of leakage, and the effect on methane air mixtures, in a gob.

e) The Application of Borehole Tests for Ventilation Planning

The techniques for determining methane content and methane desorbtion rates on borehole coal samples are now well established. It is now, necessary to better understand the meaning of these test results when coal is mined. It is probable that methane tests must be supplemented by measurements of strata pressure, gas pressure, and strata permeability. It is suggested that the Cardinal River Coals mine development is an ideal location for further testing because:

i) The borehole tests were done before mining started.
ii) The strata above and below the seam is exposed for testing.
iii) The seam is not yet developed and so provides an ideal location to observe and measure the characteristics of methane desorbtion in development headings, and methane bleed-off from developed pillars.

11.4 BRITISH COLUMBIA

a) Gas and Dust Control in Development Headings

As in Alberta, this was seen as an area for investigation. Engineers at B.C. Coal favoured the machine-mounted dust scrubber as being the most promising. However, it is possible that, in some cases, satisfactory levels of airborne dust will be attained only by a combination of methods.

The following areas for investigation are recommended:

i) The use of machine-mounted dust scrubbers.
ii) Application of high pressure water jets assisting the cutting head (see U.S.B.M. research at Bruceton, Pennsylvania).
iii) Application of overlap auxiliary ventilation and line brattice.
iv) The use of machine-mounted foam generators enabling foamed dust suppressants to be sprayed on the cutting head of continuous miners.

The fully committed collaboration of at least one operator will be an essential element of a successful investigation.

b) Pressure Balancing Across Hydraulic Mine Gob

Interest has been expressed in the development of a ventilation system enabling pressures around the gob in an hydraulic mine to be equalized sufficiently to prevent the inception of spontaneous heating. The No. 400 block of the B.C. Coal hydraulic mine was suggested as a potential investigation site.

c) Light Alloy Blades on Main and Booster Fans

There is a cost problem which faces coal mine operators relating to the material used to construct fans blades. In general, Provincial Regulations do not permit the use of aluminum and light alloy metals in underground coal mines because of the hazard of high temperature sparking. In most cases, however, they are willing to give exemptions in the case of the blades of main fans at the surface.

U.S. Regulations do not restrict the use of light alloys in coal mines and neither do the Regulations for Canadian hard-rock mines. This means that Canadian coal mine operators either must go to Europe for fans or they must pay high additional costs for steel blades when purchasing in North America.

Research should be directed towards the use of other materials in fan blades and fan casings to eliminate this hazard (See National Committee Report, 1977, p. 21). Two lines of research and development are suggested:

- i) An alternative material for the fan blades.
- ii) A special material to line the inner side of the casing that will not produce incendive sparking if struck by the blades.

11.5 GENERAL

11.5.1 Forecasting Methane for Ventilation Planning

The relationship between the results of methane desorption tests on coal samples and the observed emission rates in mines should be evaluated so that maximum use can be made of desorption test results in mine ventilation planning. In discussions with operators and those concerned with mine planning, it was evident that the work previously carried out by CANMET into determination of the total methane content in a coal seam and the rates of methane desorption was being used for ventilation planning in new mines; examples of this were at Cape Breton, in the proposals to extend the workings of No. 26 Colliery, and also in Alberta, in assessing the ventilation needs of an underground, hydraulic mine pumping station. The methane values determined by CANMET are of necessity being used theoretically. There are reservations as to whether the laboratory values will be useful as a guide to emission rates experienced when coal is mined.

It is recommended that the CANMET efforts to date should be continued in this field, particularly with reference to correlating laboratory tests with subsequent experience in the mine. It is suggested that two areas will be of particular interest:

- a) permeability of coal to methane migration, taking into account the direction of cleat, and

b) the extent of the coal zone or envelope surrounding an airway from which the methane will be desorbed. This should probably take into account the effect of air pressure and strata pressure (i.e. depth) as well as deciding the parameters used to determine when the coal can be considered drained for practical purposes.

The Cardinal River Coals' mine at Hinton, Alberta is about to enter a new coal area. Evaluations of methane content and desorption rates have already been carried out by CANMET. This might be one area where work could begin by correlating laboratory results with actual mining experience.

11.5.2 Design and Analysis of Ventilation Systems

In order to assist in more accurate ventilation planning, a comprehensive listing of friction factors should be established. These factors should be measured in Canadian Mines for a wide variety of roadway sizes, shapes and linings and for a variety of contained equipment, i.e. conveyors, etc.

An improvement in the quality of basic information regarding roadway resistances will enhance the application of computers to mine ventilation networks. It is noted that the larger coal mines have access to suitable computer terminals. It is not considered necessary for anything to be developed in terms of computer hardware. Operators should be informed of the contents and availability of computer programs dealing with ventilation so that the latest technology can be applied at the mines.

11.5.3 Ventilation Stoppings

Operators readily agree that their present stopping construction methods are not wholly satisfactory. They are not quickly constructed; they are not always fire resistant and the materials of construction are bulky. In most mines, due to strata movement, such as crushing and floor lift, stoppings are in constant need of repair and are a source of ventilation leakage.

Efforts should be directed towards:

a) The development of blocks or slabs that can be used to construct the wall of a stopping, and which will accept a substantial amount of crushing without fracturing.

b) The evaluation of sealants for application around the edges of seals.

Important features to be appraised should include:

- Mechanical form and other characteristics, e.g. solid, foam, skin type, brittle, elastic, compressible, etc.
- Resistance to burning; production of toxic fumes when burned or heated.
- Effect on persons and means of protection, i.e. toxic fumes, eye and/or skin irritant. Protective devices needed.
- Ease of cleaning applicators and associated equipment such as spray guns and mixing vessels.

A variety of stopping construction methods should be tested at selected mine sites. Labour and material costs of construction should be estimated. The application of inflatable stoppings should be investigated using alternatives to compressed air. An inflatable, nonflammable stopping could be an effective means of quickly controlling underground fires.

In general terms, Canadian coal mine ventilating efficiencies can be improved, leading to better working conditions and reduced costs. This can be best achieved by mine planners and designers giving more attention to the ventilation requirements and mine layout in the early stages and by aiming to reduce pressures and the number of stoppings required. This will require a willingness on the part of operators to try new structures and materials, including nonflammable, prefabricated, and sectionalized devices for stoppings, air-crossings, and doors.

Other materials and techniques should be tried such as the remote pumping of stoppings and seals. In this method, a central pumping station delivers a wetted fly-ash/cement mixture to the stopping site. This would be particularly applicable in short term mines driven from the outcrop where the pumping/mixing equipment could be located at the surface.

11.5.4 Methane Drainage

There have been numerous occasions in the past when mines in gasssy seams in the Rocky Mountains have experienced difficulty maintaining a safe working environment by ventilation only. If similar conditions are encountered in the future, serious consideration should be given to a trial of methane drainage systems using cross-measure boreholes, in-seam drilling, or drainage from surface drill holes. This should include pre-drainage from the solid ahead of development drivages.

11.5.5 Mining Layout

This subject is continuously under review by operators. CANMET's roles are expected to include drawing the operators attention to areas where new ideas in layout can be introduced; advising the operators and/or mine inspection bodies as to how the effectiveness of a new layout can be appraised; and actually designing or proposing a monitoring system by which the operator can measure the success of a layout in controlling methane, airborne dust, and leakage.

The development of underground mine layouts in which the main trunk conveyors are located in a roadway ventilated by air returning to the outside or to a return shaft or slope should be encouraged in mines where methane emission rates are consistently low. This has obvious advantages for preventing the airborne dust from conveyor lines from being carried into the workings. It also has an added safety feature; long coal conveyors are a notorious fire hazard. Fumes and smoke from a conveyor fire in a return airway would not constitute such a serious threat to persons working inbye.

11.5.6 Instrumentation

Increased use of devices for remote monitoring of conveyor roads, main ventilation fans and auxiliary ventilation systems should be encouraged. The certification by E.M. & R. of systems and/or instruments to perform these tasks should be pursued. At this time, the use of radio scanning of signals from sensing devices appears to be favoured by operators. A reliable system of this type is invaluable to supplement manual inspection. In addition, the development of hand-held, portable instrumentation for use during manual inspections should be examined. A single instrument or portable apparatus which can be used to make air quality determinations (as opposed to the methanometer used for methane only) would be useful. It is suggested that such an apparatus should determine:

- methane content
- oxygen sufficiency (to replace the flame lamp)
- carbon monoxide and possibly carbon dioxide.

A compact hand-held, portable, infrared heat detector for locating incipient fires and/or heatings would be a useful aid when making manual inspections.

11.5.7 Documentation of Spontaneous Heating

It is recommended that the factors related to underground spontaneous heatings, large and small, be well documented in order that lessons can be learnt for the future. Most operators already do this and, with their cooperation, it should not be too difficult to document the collective experience on a national scale. Such information could be particularly useful as new mining techniques are put into operation in Alberta's low rank Plains coals. Possibly a working party comprising CANMET engineers and coal operator's engineers could be set up for this purpose.

The documentation should include:

- district layout and ventilation,
- mining method and operating system

- detection of heating (signs and symptoms; date),
- immediate action and its result,
- longer term action and its results,
- methods of sealing (location, construction, dimensions, etc. of seals)
- observations, comments and explanations advanced by people concerned (including time durations, sample analyses, manpower employed, etc.).

11.5.8 Development of Mine Ventilation Equipment and Instrumentation in Canada

In preparing this report and drawing up some of the recommendations, the dependence of Canadian coal mines on imported equipment, systems and apparatus was noticeable. It is perhaps not too far outside the terms of reference to suggest that, if Canadian manufacturers can be encouraged to develop E.M. & R. certified equipment and instrumentation for Canadian mines, then this should reduce shipping costs, improve the Canadian balance of payments, develop Canadian technology, and possibly produce equipment acceptable to mines in other countries, which could be exported. The consultants recognize that this observation is valid only for a limited range of equipment, due to the small size of the Canadian coal industry. However the use of similar equipment in hardrock mines would, in some cases, be beneficial.

12. CONCLUDING STATEMENT

Canadian coal mines require a variety of ventilation techniques to ensure a safe environment in the wide range of underground mining conditions encountered across the country. However, some of these conditions are peculiar to Canada and not commonly found in the world of coal mining. Consequently, a significant amount of experience relevant to these unusual conditions is not available, either in the extraction of coal or the ventilation of mines. A prime example is the mining of a 13 m thick seam of friable coal dipping at 30^0 near Sparwood in British Columbia; another is the recovery of a gassy seam of submarine coal, at considerable depth, 6 km or more from the coastline of Cape Breton.

The implications for Canadian mining engineers concerned with the safe ventilation of these mines may be summarised as follows:

a) Ventilation experience in other countries, whilst useful to Canadians, will not offer much help in solving our more unusual or difficult problems.

b) As the industry expands, mechanisation becomes more sophisticated; the rate of production per unit face increases and ventilation problems are likely to become more, rather than less, critical.

c) Even though the underground coal industry in Canada is expected to expand it will not, for some time, reach a level of production or importance comparable with other leading coal producing countries. This may limit the technical resources and funds available for coal mining research and development work. Such funds and resources are available must be utilised to solve problems which, if not unique to Canada, are at least unusual.

d) Canadian mining engineers must be familiar with the state of art of modern, conventional, ventilating practice. There is however, an additional need, to be constantly recognizing and collecting information that may be helpful in defining our ventilation problems found in some of Canada's coal mining conditions.

The report prepared by the consultants will, hopefully, play a part in this strategy by presenting;

a) A summary of the state of the art in ventilation practice for a number of selected items of particular importance to Canadian mines.

b) An extensive bibliography to facilitate the study of these and other ventilation problems in more detail.

c) A description of the more critical ventilation problems faced in Canadian coal mines.

d) A series of recommendations suggesting research and development work to be implemented by research organisations such as CANMET and/or mine operators.

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POOR

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